NI 43-101 Technical Report
Apollo Gold Corporation
Black Fox Project
Timmins, Ontario, Canada

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Summary (Item 3)

SRK (US), Inc. (SRK), AMEC Americas Limited (AMEC), and Samuel Engineering, Inc (SE) were commissioned by Apollo Gold Corporation (Apollo) to prepare a Mineral Resource and Mineral Reserve Estimate compliant with National Instrument 43-101 (NI 43-101) Feasibility Study (FS) of the Black Fox open pit and underground gold project (Black Fox) in Timmins, Ontario, Canada.

The FS considered three milling options for the treatment of open pit and underground ores from Black Fox:

- Holt Mill – Toll milling up to 2,500tpd ore at St. Andrews Goldfields' (SAS) Holt Mill;
- Stock Mill – Milling up to 1,100tpd at the Stock Mill currently owned by SAS but the subject of a letter of intent signed in March 2008 between Apollo and SAS for Apollo to purchase the mill from SAS; and
- Black Fox Mill – Design build and operate a 1,500tpd mill at the Black Fox mine.

Given these alternatives, it was determined that the best option for Apollo Gold, given the current understanding of the Black Fox deposit is to operate the Stock Mill at full capacity (nominal 1,100tpd) and toll mill the remaining mine production (nominal 1,400tpd) at the Holt Mill.

The Black Fox Mill, while designed to feasibility-level, will be kept on reserve, until a time when additional reserves are discovered at Black Fox or when other, currently unknown, conditions make the construction of the Black Fox Mill economic.

Property Overview

Black Fox is located approximately 10km east of the town of Matheson, Ontario, Canada on the eastern side of the Porcupine District and approximately 75km east of the Timmins Gold Camp. The project is on the east-west trending 200km Destor-Porcupine Fault Zone (DPFZ). The mine site and its facilities are located on the south side of Highway 101 East. Supplies and services are available in Matheson or Timmins and can be delivered with a 12-hour turnaround.

The Black Fox Project is located east of the Dome Mine, now part of the Porcupine Joint Venture, located in South Porcupine near Timmins, Ontario and approximately 65km west of the Project area. Properties proximal to the Project area include the Clavos, Hislop, Holloway, Holt and Taylor all held by SAS and Ross held by Preston Electrical and Mechanical.

Temperature ranges from 20°C to 33°C during the summer months and -30°C to 10°C during the cooler winter months of October to May. The average precipitation is 873.4mm/yr and ranges between 44.5mm in February to 100.1mm in July.

The property was first explored by Dominion Gulf in 1952 and then by Hollinger in 1962. In 1988, Glimmer Mine Inc. put together the property package using a combination of crown and private lands. In 1989, Noranda Exploration Company Ltd. (Noranda) entered into a joint venture agreement with Glimmer owning 60% of the property. During their ownership, Noranda merged with Hemlo Gold Mines Inc. (Hemlo). Exall purchased the property from Hemlo in April 1996. Apollo acquired a 100% ownership in the fall of 2002 and renamed the property "Black Fox".

Today, the property includes approximately 1,225ha of land of which, 75ha are unpatented federal land, 563ha are owned by Apollo, 129ha are leased by Apollo, 332ha where Apollo has surface rights only, and 126ha where Apollo has mineral rights but no surface rights.

Environmental & Permitting

There are no environmental liabilities at the Black Fox Mine Site. A bond of C$675,000 has been given to the Ontario Ministry of Northern Development and Mines (MNDM) for site remediation for previous mining activities at the Glimmer Mine in accordance with Ontario Regulation 240/00 for Mine Development and Closure under Part VII of the Ontario Mining Act.

The Black Fox Project currently is permitted under the following approvals:

- Certificate of Approval (C of A) for Industrial Sewage Works 4-0125-96-006;
- Amended Certificate of Approval – Air – (mine heaters and generators) 3505-56R2JP;
- Amended Certificate of Approval – Air – (laboratory) 3505-56R2JP;
- Permit to Take Water (PTTW) – (mine dewatering) 00-P-6025; and
- Waste Generator Registration ON2142400.

Geology

The Black Fox property is located within Precambrian age metavolcanics and metasedimentary rocks of the Abitibi Greenstone Belt. This is one of the world's largest Archean greenstone belts believed to have formed by a complex history of paired arc volcanism and back arc sediments subsequently deformed during continental collision. The area hosts five main rock groups, most of which have tectonic contacts of varying intensity. These include:

- Blake River Group;
- Kinojevis Group;
- Stoughton-Roquemaure Group (Black Fox Host Units);
- Hunter Mine Group; and
- Porcupine Group.

Surface, underground and exploration drilling has delineated five major rock types in the vicinity of the Black Fox mineralization. These include:

- Mafic volcanic units;
- Metasediments;
- Green carbonate schist;
- Ultramafic volcanics; and
- Felsic intrusive units.

The Black Fox mineralization is an Archean age, lode gold deposit located within the Abitibi greenstone belt. The characteristics of this deposit type include; greenstone host rocks and gold-bearing quartz-carbonate veins. The veins occur as two main types. The first are arrays and stockworks along faults and shear-zones with a quartz-carbonate laminated fault-fill. The second are widely distributed extensional veins within carbonatized metamorphosed greenstone rocks. These deposits are typically associated with crustal scale compressional faults with a vertical extent of ≤2km and limited metallic zoning (Dubé and Geosselin, 2007).

The Black Fox deposit lies along the DPFZ, a major, east-west trending, deep-seated, crustal fault zone. The DPFZ and its numerous splays are associated with many past and current producing gold mines and gold deposits in the Porcupine Camp. The Stock and Aquarius gold deposits are located immediately west of Black Fox and the Holloway and Holt-McDermott Mines are located immediately to the east. Each of these deposits hosts approximately 800k to 1Moz-Au. The Black Fox deposit is situated midway between two major mines, the Dome-Hoyle Pond and the Holt-Holloway. The Dome-Hoyle Pond deposits located within the same structural regime 65km west, have shown that gold bearing structures can be traced to 1,600m below surface where they remain open at depth. The Holt-Holloway Mine, located approximately 45km to the east has been developed down to 1,200m below surface.

There are several different styles of mineralization in the deposits associated with the DPFZ. The gold mineralization is structurally controlled, in a variety of geological settings. Alteration types include pyritic ankerite-sericite ± silica-albite altered mafic volcanics, green carbonate fuchsitic altered ultramafic volcanics with quartz stockworks, pyritic, porphyritic to syenitic felsic intrusives and multiple stages of quartz veins with free gold. Much of this variation is found at Black Fox.

Mineralization

Gold mineralization at Black Fox occurs mainly within an ankerite alteration zone 1km along strike and 20m to 100m wide. This alteration envelope occurs primarily within komatiitic ultramafics and lesser mafic volcanics within the outer boundaries of the DPFZ. In some areas, the auriferous zones occur as concordant zones, which follow lithological contacts and have been subsequently deformed to slightly discordant zones that are associated with syenitic sills. Other auriferous zones occur in quartz veins and stockworks discordant to lithology (Hoxha and James, 2007).

The three main styles of gold mineralization observed at Black Fox are:

- Low-sulfide mineralization associated with abundant quartz veining and quartz stockwork within strong ankerite-fuchsite altered ultramafic volcanic rocks;
- Mineralization hosted within mafic volcanic units associated with >5% pyrite and minor to moderate quartz veining; and
- Mineralization hosted by silicified felsic dikes.

Lakefield conducted comprehensive bench scale testwork in 1996, followed by a combination of pilot plant studies and related bench scale tests in 1999. Metallurgical testwork performed by demonstrated the Black Fox mineralization to be free-milling and devoid of deleterious elements that could adversely affect the environment or the process. Test results indicated the potential value in deploying a gravity concentration circuit. The program determined the optimum grinds for the West and East Zones to be K_{80} 50µm and K_{80} 30µm respectively. The leach kinetics were found to be most favorable, with 30 hours of leach time being sufficient to achieve optimum results.

The main conclusions developed by the Lakefield work are:

- The gold mineralization is readily amenable to cyanidation. When grinding in a sodium cyanide solution, approximately 90% of the gold contained in the mill feed is dissolved by the time the pulp has exited the cyclone overflow;
- The degree of dissolution is dependent on the leach feed grind. Optimum size distribution for west zone ore appears to be 50µm while the East Zone mineralization requires grinding 30 to 40µm;
- The Bond Ball Mill work index of the ore varies within the range of 14 to 17kWh/t;
- Gold dissolution is relatively insensitive to variations in leach times over the ranges examined;
- Black Fox mineralization contains no deleterious elements that could adversely affect operating efficiencies or the environment;
- To varying degrees, Black Fox mineralization is amenable to gravity concentration; and
- The ground mineralization exhibits favorable settling characteristics.

Resources & Reserves

The Black Fox deposit has been estimated using a modern block modeling technique. This included proper geologic input, appropriate block model cell size, assay compositing and reasonable interpolation parameters. The results have been validated using three methods including; on screen proofing, rectification to historic production and statistical comparisons between the estimated block grades and the composites used to assign them.

The Mineral Resources are classified under the categories of Measured, Indicated and Inferred Mineral Resources according to CIM guidelines. Classification of the Resources reflects the relative confidence of the grade estimates, as a function of many factors including primarily; assay data quality, QA/QC procedures, quality of density data, and sample spacing relative to geological and geo-statistical observations regarding the continuity of mineralization.

The tonnage and grade for Indicated and Inferred Resources differentiated by different mining methods at appropriate Au cut-offs are shown in Tables 1 and 2.

Table 1: Black Fox Indicated Resource Statement

Mining Method*	Category	Cut-off gpt-Au	Mt	Grade gpt-Au	Contained oz-Au (000's)
Open Pit	Indicated	1.0	4.8	5.3	813.1
Underground	Indicated	3.0	1.7	11.4	622.6

* Mining Method is determined by relative location above or below the 9,814.5m elevation

Table 2: Black Fox Inferred Resource Statement

Mining Method*	Category	Cut-off gpt-Au	Tonnes	Grade gpt-Au	Contained oz-Au (000's)
Open Pit	Inferred	1.0	2.7	4.7	408.3
Underground	Inferred	3.0	0.8	13.1	329.0

* Mining Method is determined by relative location above or below the 9,814.5m elevation

The orientation, proximity to the surface, and geological controls of the Black Fox ore body will require mining of the ore reserves with open pit and underground mining techniques. Hence, ore reserves shown in Table 3 have been subdivided into open pit and underground categories.

Table 3: Open Pit and Underground Ore Reserve Statement

Classification	Category	Resource (kt)	Grade (gpt)	Gold (koz)
Open Pit	Proven	0	0	0
	Probable	4,350	5.2	730
	Proven and Probable	4,350	5.2	730
Underground	Proven	0	0	0
	Probable	2,110	8.8	600
	Proven and Probable	2,110	8.8	600
Combined	**Total Proven**	**0**	**0**	**0**
	Total Probable	**6,460**	**8.8**	**1,330**
	Total Proven and Probable	**6,460**	**6.4**	**1,330**

These reserves are based on a gold price of US$650/oz. A cut-off grade of 0.9gpt is used in the open pit and 3.0gpt in the underground design.

Mining

The open pit operation designed for a 1,500tpd throughput will primarily use 4m^3 and 10m^3 hydraulic excavators loading 105t haul trucks. A 6.5m^3 front end loader will be used as backup to the excavators and will also be used for loading the backfill for the underground operations. Two drills will be purchased and a third one leased later in the project. The major support equipment will include two dozers, a grader and a water truck. The average stripping ratio based on the mine production schedule is 13:1 (waste to ore) with 4.4Mt or ore grading at 5.21gpt gold.

Black Fox underground mining will incorporate cut and fill mining, utilizing a mining cross-section of 3m high x 6m wide for the cut and a cemented rock fill (CRF) backfill material. Cut and fill was selected due the versatility of the method to allow the minimal amount of dilution while, meeting the production throughput target of 1,000tpd. Ore from underground will total 2.1Mt at an average grade of 8.8gpt.

Processing

The Stock Mill, which will be owned and operated by Apollo, has the capability of a throughput of 1,100tpd (396,000tpy) on ore to be processed from the Black Fox mineralization. Metsim computer simulations of the grinding circuit have determined that a series circuit, utilizing an existing third grinding mill along with reconfigured existing cyclones, will produce the optimal grind size at the rate necessary to achieve 1,100tpd. The recovery projections for the Stock mill are 95%, which is considered achievable at the given grind size of P_{80} = 55µm and leach times in excess of 24 hours. Historic production for the period 1997 to 2000 show recoveries actually achieved as 97.14%.

The Stock Mill includes the conventional unit processes:

- Primary crushing;
- Closed circuit, single staged fine crushing;
- Two staged grinding;

- Pre-leach thickener and carbon columns;
- Leach and Carbon-in-Pulp (CIP) circuits;
- Carbon stripping and electrowinning; and
- Cyanide destruction.

The Holt Mill is owned by SAS. An agreement is in place to toll-treat Black Fox ores at this facility. The Holt Mill has a nominal 3,000tpd capacity. The agreement calls for the treatment of treat up to 1,400tpd of Black Fox ores. Its flowsheet and characteristics are similar to the Stock Mill, and similar metallurgical recoveries will occur.

Tailings Facility at Stock Mill

Approximately 3.4Mt of ore from Black Fox will be processed at the Stock Mill. Golder Associates Ltd. has recently prepared a pre-feasibility report entitled "Conceptual Design of Phase 4 and 5 Raises, Tailings Management Facility, Stock Mine, Ontario". AMEC conducted a preliminary review of the design report. As some of the important information required to complete the review is not available currently, the design background data and accuracy of the design study cannot be confirmed. The most significant issues involved include:

- Absence of static liquefaction assessment of tailings, which is a key issue in tailings dams raised by upstream method of construction;
- Lack of base filter drains or filter blanket to control phreatic surface within the dam (thus, the phreatic surface is likely either to emerge on the dam slope surface or be very close to it); and
- The design strength values used for the existing and future (improved by preloading) appear to be high (in comparison to the N-values mentioned in the report) for sensitive silty clays that exist in the general area (site specific strength and sensitivity data is not available).

To advance the study to a feasibility level, Apollo has commissioned AMEC to carry out a detailed evaluation of the current concept or develop a new concept for the tailings management at the Stock Mine site and this study is currently in progress.

Capital & Operating Costs

LoM capital costs totaling US$156.1million are summarized in Table 4. Details supporting this estimate are discussed in this section. Preproduction capital costs are US$86.9million. Ongoing capital accounts for the remaining mine life. Capital cost estimates are in Q1 2008 US constant dollar terms.

Table 4: LoM Capital Costs (US$000s)

Description	Pre-Production Capital (through Q4-2008)	Ongoing Capital (Q1-2009 to End)	Total Capital
Equipment			
Open Pit Mine	$13,462	$0	**$13,462**
Underground Mine	$7,729	$4,826	**$12,555**
Infrastructure @ Black Fox	$22,989	$1,656	**$24,644**
Black Fox Mill	$0	$0	**$0**
Tailings @ Black Fox	$0	$0	**$0**
Stock Mill	$1,301	$0	**$1,301**
Tailings @Stock	$1,518	$1,432	**$2,950**
Mine Closure	$0	$16,091	**$16,091**
Owner Costs	$25,779	$1,336	**$27,116**
Equipment	**$72,777**	**$25,341**	**$98,118**
Development			
Capitalized Operating Cost	$5,732	$0	**$5,632**
Glacial Till Removal	$4,013	$16,048	**$20,061**
Underground Mined Development	$4,397	$27,774	**$32,171**
Development	**$14,142**	**$43,821**	**$57,963**
TOTAL CAPITAL	**$86,919**	**$69,162**	**$156,081**

LoM operating costs are summarized in Table 5. Operating cost estimates are in Q1 2008 US constant dollar terms.

Table 5: Cash Operating Cost Summary

Description	LoM Total (US$000s)	Unit Cost (US$/total-t)	Unit Cost (US$/ore-t)*
Open Pit Mining	$107,348	$2.13/t	$24.68/t
Underground Mining	$117,083	-	55.39/t
Mine G&A	$42,576	-	$0.81/t
Holt Toll Mill	$120,256	-	$39.54/t
Stock Mill	$84,265	-	$24.63/t
Black Fox Mill	$0	-	$0.00/t
G&A	$19,907	-	$3.08/t
Refining, Transportation & Insurance	$1,615	-	$0.25/t
Total	**$487,319**	**-**	**$75.40/t** **$386.57/Au-oz**

*Weighted average over the LoM.

Technical-Economic Results

The technical-economic results are based upon work performed by Apollo's engineers and consultants. All costs are in Q1 2008 US constant dollars. The economic model is pre-tax and assumes 100% equity to provide a clear picture of the technical merits of the project. The LoM plan and economics are based on the following:

- A gold price of US$750/oz;
- Probable reserves, no resources are included;
- A mine life of 8.75 years, at a designed rate of 875ktpy;
- An overall average metallurgical recovery rate of 95% Au, over the LoM;
- A cash operating cost of US$75.40/t-milled, US$386.57/oz-Au;

- Initial capital costs of US$86.9million. LoM capital costs are estimated to be US$156.1million being comprised of US$58.0million for capitalized development and US$98.1million for mine equipment;
- Mine closure cost is US$16.1million; and
- No salvage value is modeled.

The base case economic analysis results, shown in Table 6, indicate a pre-tax net present value of US$227.1million at a 5% discount rate with an IRR of 62%.

Sensitivity analysis for key economic parameters are shown below in Table 7. This analysis suggests that the project is most sensitive to market price. Operating costs are slightly more sensitive than capital costs due to the many operating functions associated with the project. Also, the purchase of the existing Stock Mill resulted in a lower than 'typical' capital cost for this project, which has the effect of making capital costs less sensitive.

Table 6: Technical Economic Results ($000s)

Description	Technical Input or Result
Ore	
Open Pit	
Waste	56,881kt
Ore	4,350kt
Total	**61,231kt**
s/r	13.0
Grade	5.218gpt-Au
Contained Gold	729koz
Underground	
Total Development	35,920m
Ore	2,114kt
Grade	8.82gpt-Au
Contained Gold	599koz
Mill	
Ore Treated	
Holt Toll Mill	3,116kt
Stock Mill	3,348kt
Black Fox Mill	0kt
Total	**6,464kt**
Ore Grade	
Holt Toll Mill	6.30gpt-Au
Stock Mill	6.47gpt-Au
Black Fox Mill	0.00gpt-Au
Total	**6.39gpt-Au**
Contained Gold	
Holt Toll Mill	628koz
Stock Mill	700koz
Black Fox Mill	0koz
Total	**1,328koz**
Recovered Gold	
Holt Toll Mill	603koz
Stock Mill	665koz
Black Fox Mill	0koz
Total	**1,268koz**
Revenue ($000s)	
Gross Revenue	**$945,455**
Refining & Transportation Charges	$1,615
Net Smelter Return	**$943,840**
Royalty	$0
Gross Income From Mining	**$943,840**
Realized Price (Gold)	*US$748.72/oz-Au*
Operating Cost ($000s)	
Open Pit Mine	($107,348)
Underground Mine	($117,083)
Mine G&A	($42,576)
Holt Toll Mill	($120,256)
Stock Mill	($84,265)
Black Fox Mill	$0
G&A	($19,907)
Operating Costs	**($485,704)**
	US$385.29/oz-Au
	US$75.15/t-milled
Cash Operating Margin	**$458,136**
	US$363.43/oz-Au
	US$70.88/t-milled
Capital Cost	
Equipment	($98,118)
Development (Capitalized)	($57,963)
Total Capital	**($156,081)**
Cash Flow	**$302,055**
(NPV$_{5\%}$)	**$227,081**
IRR	**62%**

Table 7: Project Sensitivity ($NPV_{5\%}$, $000's)

Description	-10%	-5%	Base Case	+5%	+10%
Gold Price	$152,715	$189,898	**$227,081**	$264,264	$301,447
Operating Costs	$266,356	$246,719	**$227,081**	$207,443	$187,806
Capital Costs	$235,932	$231,236	**$227,081**	$222,926	$218,770

Conclusions

The Black Fox deposit has been adequately drill tested to estimate grade and tonnes classified as Indicated and Inferred Resources. The estimation results have defined an Indicated Resource potentially exploitable by open pit mining at a 1gpt-Au cut-off, of 4.8Mt with an average grade of 5.3gpt-Au containing 0.8Moz of gold. Additionally, it defines an Indicated Resource potentially exploitable by underground mining at a 3gpt-Au cut-off, of 1.7Mt with an average grade of 11.4gpt-Au containing 0.6Moz of gold.

The open pit and underground mine design have defined a combined Indicated ore reserve of 6.5Mt at 6.4gpt-Au for 1.3Moz of gold.

The Feasibility Study demonstrates that the project is technically feasible and has a robust economic performance with the design and operating criteria used and the assumed gold price projections.

A key factor to the robust economic performance is the recent agreement to acquire the existing Stock Mill at a substantial discount to the cost of a new mill, notwithstanding the requisite permitting and equipment lead times for the construction of a new mill.

Recommendations

Black Fox should continue to be developed to the detailed engineering level. The following recommendations for the project should be considered by Apollo:

- Continue to core drill specific areas of the ore body to further upgrade and extend the geological modeling for the project;
- Complete Stock Mill tailings testwork;
- Establish optimal Stock Mill capacity;
- Complete detailed engineering design work in all areas; and
- Refine the project implementation schedule.

Estimated cost for these recommendations is US$3.0million.

1 Introduction (Item 4)

1.1 Scope of Work and Terms of Reference

SRK (US), Inc. (SRK), AMEC Americas Limited (AMEC), and Samuel Engineering, Inc (SE) were commissioned by Apollo Gold Corporation (Apollo) to prepare a Mineral Resource and Mineral Reserve Estimate compliant with National Instrument 43-101 (NI 43-101) Feasibility Study (FS) of the Black Fox open pit and underground gold project (Black Fox) in Timmins, Ontario, Canada. Black Fox is located approximately 10km east of the town of Matheson, Ontario, Canada along the east-west trending 200km Destor-Porcupine Fault Zone (DPFZ).

The Glimmer underground gold mine operated on the Black Fox property over the period 1997-2001, and produced approximately 211koz of gold by contract milling in either the St. Andrew or Macassa mills. Underground mining extended to depths of approximately 200m to 215m below the surface before operations were suspended due to low gold prices in May of 2001.

Apollo purchased the property from the Exall-Glimmer joint venture in 2002 and began exploration of the property in 2003. The Apollo exploration drilling programs have intersected significant gold mineralization in both near surface, and down-dip of the area mined by Exall Resources Ltd. (Exall), as well as along strike.

This FS is intended for the use of Apollo for the further development and advancement of Black Fox to the production stage. This report meets the requirements for NI 43-101 and the Resource and Reserves definitions are as set forth in the Appendix to Companion Policy 43-101CP, Canadian Institute of Mining, Metallurgy and Petroleum (CIM) – Definitions Adopted by CIM Council, November 2005.

1.2 Qualifications of the Consultants

This FS has been prepared by a team of consultants sourced principally from SRK's Denver, Colorado office, AMEC's Mississauga, Ontario office, and SE's Denver office (the Consultants). These consultants are specialists in the fields of geology, exploration, Mineral Resource and Mineral Reserve estimation and classification, open pit mining, underground mining, geotechnical, environmental, permitting, mineral processing and mineral economics disciplines.

Neither the Consultants, their employees or associates employed in the preparation of this report have any beneficial interest in Apollo. The Consultants will be paid a fee for this work in accordance with normal professional consulting practice.

The individuals who have provided input to this FS have extensive experience in the mining industry and are members in good standing of appropriate professional institutions. The key project personnel contributing to this report are listed in Section 1.2.1.

1.2.1 Project Team and Responsibilities

The Qualified Persons (QPs) for this report are listed in Table 1.2.1.1. A brief description of key contributors to the Project is provided below.

Table 1.2.1.1: Key Project Personnel

Company	Name	Discipline
SRK	Bart Stryhas CPG, PhD	QP, Overall Reporting, Resource Estimation, Site Visit
	Dorinda Bair BS Geo	Geology
	Bret Swanson, B.E. Mining	Open Pit Mining and Reserves
	Martin Raffield, P.Eng., PhD	QP, Mining, Reserve Estimate, Site Visit
	David Young, P.E.	UG Mining and Reserves
	Nick Michael BS Mining, MBA	Project Economics
Apollo	Ryan Lougheed, BS Environmental	Environmental
	Jeff Choquette, BS Mining	Open Pit Mining
	Bill Rust, BS Metallurgy	Metallurgy
AMEC		
	Debbie Dyck, BASc, PEng	QP, Social and Environmental
	Xiaogang Hu, PhD	QP, Geotechnical and Water Management
SE		
	George Burgermeister, BS Metallurgy	Metallurgy, Processing and Infrastructure
	Randolph P. Schneider, MAusIMM	QP, Metallurgy, Processing and Infrastructure

1.3 Reliance on Other Experts (Item 5)

The Consultant's opinions contained herein are based on information provided to them by Apollo throughout the course of their investigations. The sources of information include data and reports supplied by Apollo personnel, as well as documents listed in Section 20.

The Qualified Persons preparing and supervising this FS have not relied on a report, opinion or statement of a legal or other expert, who is not a Qualified Person for information concerning legal, environmental, political or other issues and factors relevant to this FS.

The Consultants used their experience to determine if the information from previous reports was suitable for inclusion in this FS and adjusted information that required amending. Revisions to previous data were based on research, recalculations and information of each of the Qualified Persons. The level of detail utilized was appropriate for this level of study.

This report includes technical information, which requires subsequent calculations to derive sub-totals, totals, and weighted averages. Such calculations inherently involve a degree of rounding and consequently can introduce a margin of error. Where these rounding errors occur, the Consultants do not consider them to be material.

2 Property Description and Location (Item 6)

2.1 Property Location

Black Fox is located 10km east of Matheson, Ontario, along Hwy 101 East and approximately 655km north of Toronto, Ontario. It is located in the Hislop and Beatty townships, District of Cochrane, in the Larder Lake Mining District 90. The project is centered at 48°30'N latitude and 80°21'W longitude. The Glimmer underground mine, formerly operated by Exall is located within the property boundaries. Figures 2-1 and 2-2 show the property location (Prenn, 2006).

2.2 Land Area

The property includes approximately 1,225ha of land of which, 75ha are unpatented federal land, 563ha are owned by Apollo, 129ha are leased by Apollo, 332ha where Apollo has surface rights only, and 126ha where Apollo has mineral rights but no surface rights. Figure 2-3 shows land tenure for Black Fox and Table 2.2.1 lists the current land position for the Project.

Table 2.2.1: Current Black Fox Project Property Summary

Township	Concessions	Lot #	Parcel	Acreage	Hectares	Status
Hislop	6	4	3393	81	33	Owned by Apollo
Hislop	6	5	11511	160	65	Owned by Apollo
Hislop	6	6	6413	82	33	Owned by Apollo
Hislop	6	6	2582	161	65	Owned by Apollo
Hislop	6	7	388	147	59	Owned by Apollo
Hislop	6	7	15466	97	39	Owned by Apollo
Hislop	6	7	4707	46	19	Owned by Apollo
Hislop	6	8	7745	164	66	Owned by Apollo
Beatty	1	6	14572	152	61	Owned by Apollo
Beatty	1	5	24577	156	63	Owned by Apollo
Beatty	1	8	4150	148	60	Owned by Apollo
Subtotal				*1,392*	*563*	*Owned*
Hislop	4	3	16262	159	64	Leased by Apollo
Hislop	4	2	16265	80	32	Leased by Apollo
Hislop	4	2	16266	80	32	Leased by Apollo
Subtotal				*319*	*129*	*Leased*
Beatty	1	5	15639	39	16	Surface Rights
Beatty	1	5	15653	39	16	Surface Rights
Beatty	1	5	15636	39	16	Surface Rights
Beatty	1	5	15651	39	16	Surface Rights
Beatty	1	5	15652	39	16	Surface Rights
Beatty	1	5	15670	39	16	Surface Rights
Beatty	1	5	14576	39	16	Surface Rights
Beatty	1	5	14567	39	16	Surface Rights
Beatty	1	5	15669	39	16	Surface Rights
Beatty	1	5	15662	39	16	Surface Rights
Beatty	1	5	15660	39	16	Surface Rights
Hislop	5	3	10255	79	32	Surface Rights
Hislop	5	4	24023	68	28	Surface Rights
Hislop	5	4	23687	79	32	Surface Rights
Hislop	5	5	23687	161	65	Surface Rights
Subtotal				*819*	*332*	*Surface Rights*
Beatty	1	7	3524	156	63	Mineral Only
Beatty	1	7	11720	156	63	Mineral Only
Subtotal				*311*	*126*	*Mineral Only*
Beatty	1	6	L-1115059	41	17	Claim
Hislop	6	5	L-1048333	41	17	Claim
Hislop	6	5	L-1048334	41	17	Claim
Hislop	6	7	L-1113087	22	9	Claim
Hislop	6	6	L-1048335	41	17	Claim
Subtotal				*185*	*75*	*Unpatented*
Total Land				**3,026**	**1,225**	**All Types**

2.3 Mining Claim Description

All of the Black Fox claims are current and the required claim fees and work commitments have been completed. All of the claim corners have been surveyed.

Apollo has kept all of their claims current and up to date as far as fee and work commitments. A settlement agreement dealing with a dispute over control of four claims that were staked by a party (Respondents) who believed that Apollo allowed their rights to the claim to expire has been settled. With this agreement, the Respondents are surrendering present and future rights to this property and places the property in question under the control of Apollo for both surface and mineral rights. Apollo has applied to the Ministry of Northern Development and Mines (MNDM) to bring these four claims to lease. The application fee and first year's rental of C$3/ha were sent on March 24, 2008 and Apollo expects approval of this application. The duration of the lease will be 21 years for mineral and surface rights on claims L1048334, L1048335, and L1115059 and 21 years for mineral rights only for L1113087.

A tentative agreement has been reached with a third party staking interest on a fifth claim and is expected to be closed on April 17, 2008. Apollo has filed the initial application and plan survey with MNDM. MNDM has accepted the initial application and will approve it once the agreement is completed. All paperwork regarding this agreement should be completed by April 30, 2008 and the property should be brought to lease by Apollo no later than June 30, 2008.

2.4 Agreements and Encumbrance

As described in Section 2.1., Black Fox was purchased by Apollo from the Glimmer Mine Joint Venture. The Glimmer Mine Joint Venture included Exall and Glimmer Resources. The Glimmer Mine (now Black Fox) was transferred on September 7, 2002. As of that date, Apollo owns 100% of Black Fox, which includes all existing infrastructure and buildings on the property. The Glimmer Mine Joint Venture was paid C$3,159,000 and 2.08M shares of Apollo stock. An additional cash payment of C$3million was made in January 2006.

The Project is currently advanced exploration status. However, Apollo submitted a Notice of Project Status for the Project, which is required by Section 141.(1)(a) of the Mining Act, R.S.O., Chapter M14 for Mine Production, on April 17, 2007. This notice has subsequently been circulated by the MNDM to the Ministries of Environment, Labour and Natural Resources, the Township of Black River Matheson, Department of Fisheries and Oceans and Environment Canada for their information. Since a Closure Plan for Mine Production has been filed for this site, MNDM requires Apollo as per Section 141.(1)(b) and (c) of the Mining Act, to provide public notice and file a new certified Closure Plan prior to commencement of mine production. In addition, Financial Assurance must be submitted to the MNDM with the Closure Plan.

2.4.1 Stakeholders and Interested Parties

Stakeholders with authority of some nature at the property will include Apollo, the regulatory agencies, the general public, and non-government organizations (NGOs). Other Stakeholders include the First Nations of the Abitibi Indian Reserve 70, which is jointly owned by the Abitibiwinn (Quebec) and Wahgoshig (Ontario) First Nations, and local private landowners in both Hislop and Beatty Townships. The Abitibi Indian Reserve 70 is located 25km east of the mine site. Table 2.4.1.1 lists the local private landowners described as stakeholders near Black Fox (Dyck, 2007). Any adjoining property not listed in Table 2.4.1.1 is crown land.

Table 2.4.1.1: Local Stakeholders

Land Description	Stakeholder
Hislop Township	
Parcel # 16617	Paul and Christine Bagordo
Parcel # 4184	Winston and Diana Plant
Parcel # 9385	Winston and Diana Plant
Parcel #1365	Ray Durham
Parcel #10706	Ed Shannon
Resident on Concession 6, Lot 7	John and Gloria Barber
Resident on Concession 6, Lot 6	Winston and Diana Plant
Beatty Township	
Parcel # 3265	1051989 Ont. Inc.
Parcel # 15661	Timmins Forest Products
Residents on Concession 1 Lot 7	George and Evelyn Truax; Joe and Margaret Patterson; Gerald Shannon
Parcel # 23723	Jalbert Logging

Source: Dyck, 2007

2.5 Environmental Liabilities

There are no environmental liabilities at the Black Fox Mine Site. A bond of C$675,000 has been given to the Ontario Ministry of Northern Development and Mines (MNDM) for site remediation for previous mining activities at the Glimmer Mine in accordance with Ontario Regulation 240/00 for Mine Development and Closure under Part VII of the Ontario Mining Act.

2.6 Permitting and Completed Studies

2.6.1 Permitting

"In December 2004, Apollo submitted a Closure Plan for the existing conditions of the Black Fox Project site, which supersedes the unaccepted Hislop-Beatty Project Closure Plan that was previously submitted by Exall in 1996. A new Closure Plan will be developed for the proposed future development of the Project, described herein, in compliance with legislation and directives from all pertinent regulatory bodies.

The Black Fox Project currently is permitted under the following approvals:

- Certificate of Approval (C of A) for Industrial Sewage Works 4-0125-96-006;
- Amended Certificate of Approval – Air – (mine heaters and generators) 3505-56R2JP;
- Amended Certificate of Approval – Air – (laboratory) 3505-56R2JP;
- Permit to Take Water (PTTW) – (mine dewatering) 00-P-6025; and
- Waste Generator Registration ON2142400.

Upon obtaining the property, Apollo has undertaken to clarify historical permits and obtain new permits required by new or amended legislation.

2.6.2 Completed Engineering and Environmental Studies

Terrestrial Environment

Limited terrestrial investigations in the site area were conducted in 1994 and 1995 by AMEC (formerly AGRA, for another mining client). Studies were conducted by Beak in 1996, which focused on wetlands along Salve Creek and the shoreline of Froome Lake. Additional investigations were conducted by AMEC in 2005 to supplement the findings of both the AMEC and Beak studies in areas surrounding the mine site, Salve Creek and Froome Lake. As well, additional surveys have been undertaken to provide further details on terrestrial vegetation and wildlife in areas that may be affected by future mining activity, such as in the vicinity of the overburden and waste rock storage piles. Depending on the final detailed designs, additional studies may be undertaken.

Hydrogeological Characterization

A number of investigations have been completed to support the characterization of the groundwater regime in the vicinity of the Black Fox Project. Pump tests on large diameter wells, and monitoring of groundwater levels in exploration holes, were conducted in order to determine the characteristics of the overburden aquifer. Packer testing was also completed on a select number of diamond drill exploration holes to estimate bedrock permeability. The data obtained during these tests has been used to estimate the amount of groundwater that would potentially report to the open pit from the overburden aquifer.

Additional testing has been conducted on selected wells to help approximate in-situ hydraulic conductivity values for each screened interval. A three dimensional, conceptual groundwater model has been developed using the field data obtained to predict the potential effects of mine development activities on the local groundwater and surface waters (e.g., drawdown effects).

Hydrological and Aquatic Habitat Assessments

Hydrological assessments in the past were in large part developed by pro-rating regional flow data to the local watershed areas. Current studies are focusing on developing more accurate estimates of stream flows, runoff volumes and site drainage patterns associated with the existing mine site and future developments. Efforts include detailed watershed mapping initiatives, as well as the development of a stream flow monitoring station on the Pike River and a water level monitoring station in Froome Lake. This information will be crucial in assessing potential adverse environmental effects to the downstream aquatic receiving environment and assisting in storm water management planning activities.

Aquatic habitat assessments undertaken in 2004 were based on data collection initiatives recommended in prior studies (Beak), in the context of the proposed project, and additional sampling of stream and lake sediments, water chemistry and benthic macro invertebrates were also undertaken. As well, future aquatic assessment programs will be expanded to include areas that could potentially be affected by future mining activity. Of particular importance is the thorough assessment of potential fisheries habitat areas in the areas of proposed mine development.

Geotechnical Considerations

A geotechnical investigation program was conducted in support of the project development activities. The program focused on the following areas of development: 1) the open pit perimeter slopes (overburden only), 2) potential site buildings and access road foundations, 3) the overburden stockpile, 4) the waste rock stockpiles, and 5) the tailings impoundment. Subsequent design stages will include a more extensive field program at the exact locations of the structures, and additional engineering analyses.

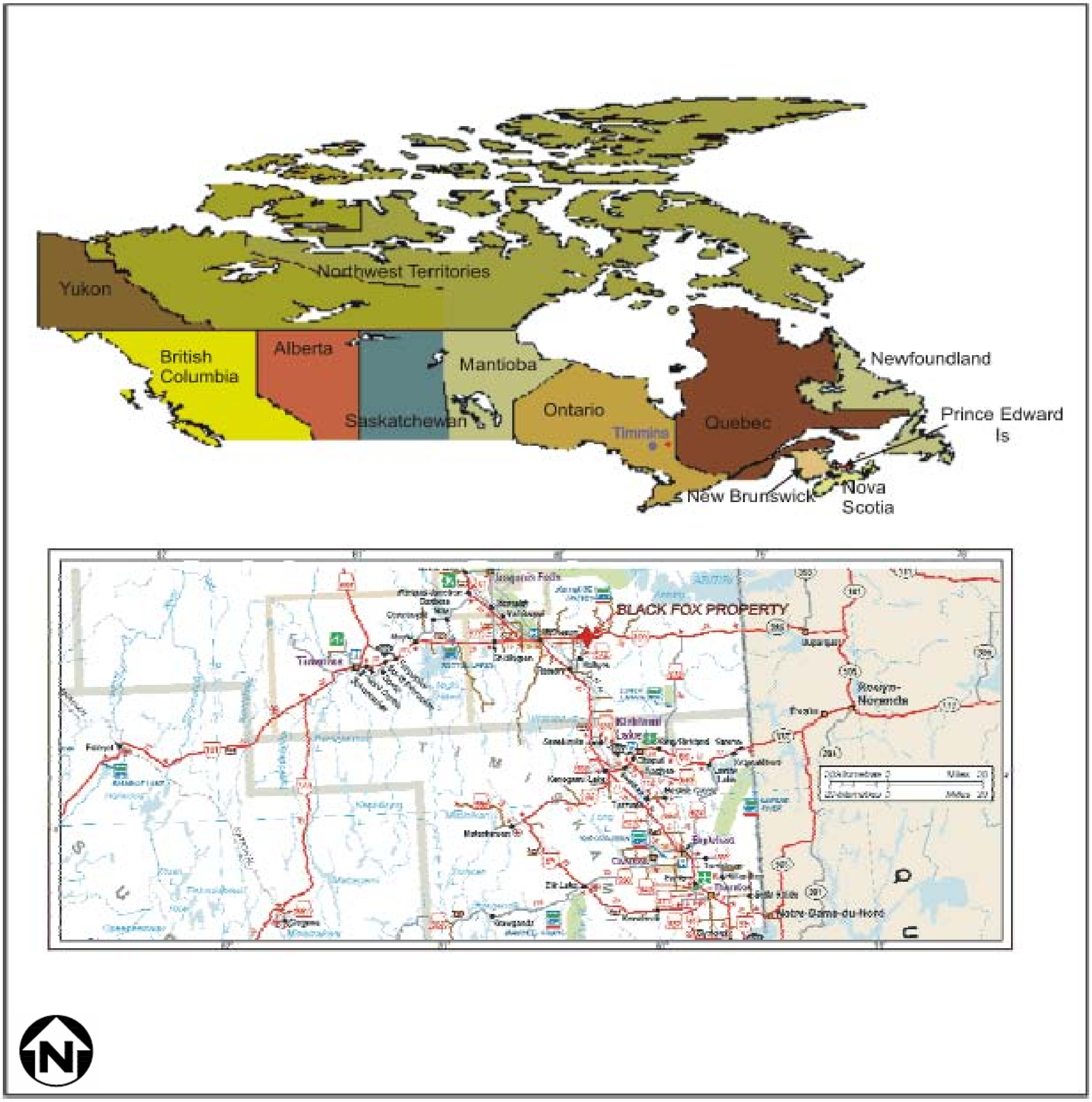
With respect to foundations for buildings and other structures, site services and access roads, geotechnical investigations have been designed to develop preliminary recommendations for potential foundation types, assess available bearing capacities at certain founding elevations, and to better understand expected excavation conditions, bedding requirements for services and access road granular thickness design.

Planning for the excavation of the overburden in the pit area (pit stripping) will have to consider the type of overburden, the location of the groundwater table, safe slope configurations, as well as run-off collection and management.

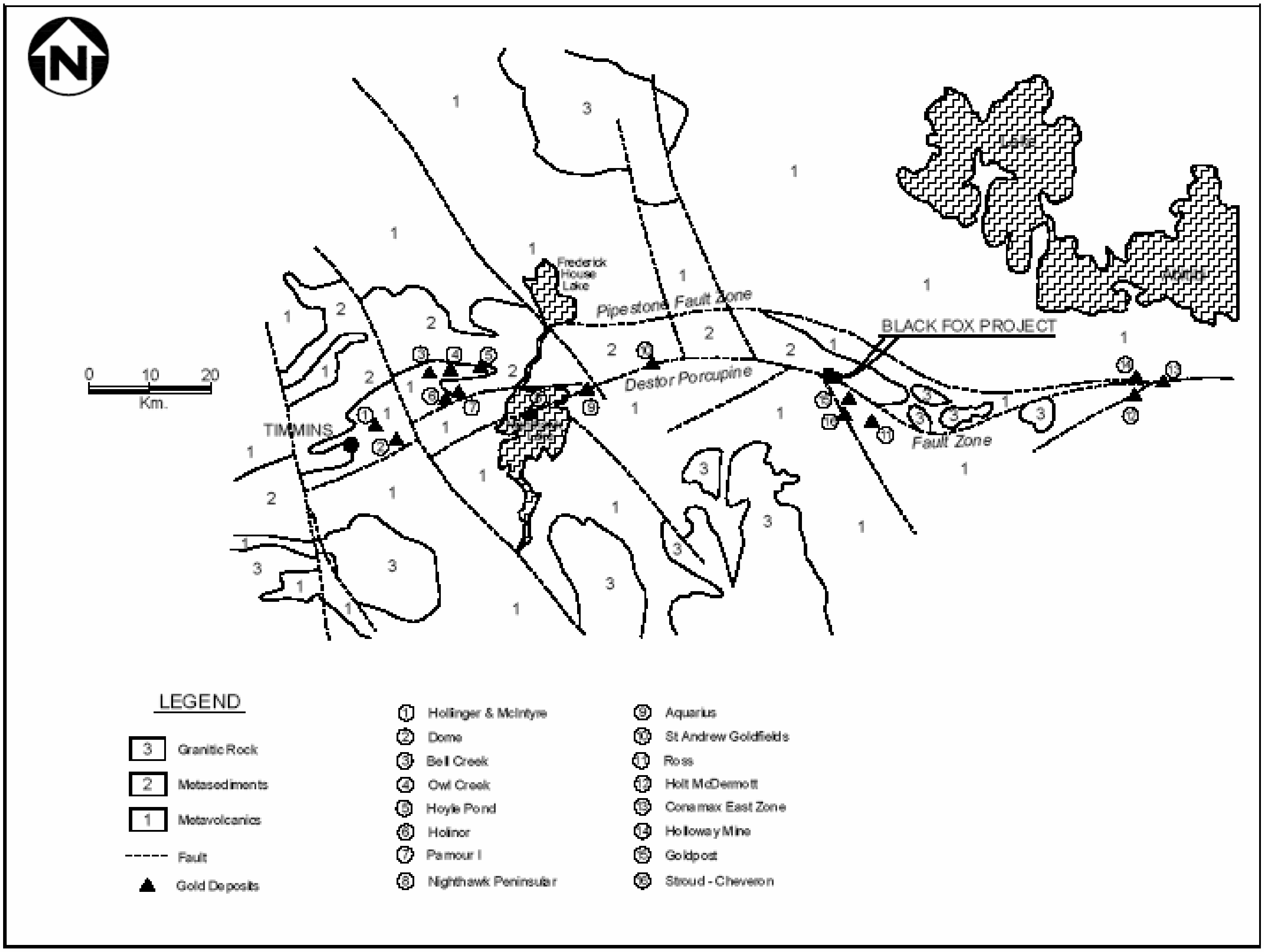
Geotechnical investigations in the vicinity of the proposed waste rock and overburden piles were intended to assess and clarify potential subgrade preparations for the placement of material to ensure long term stability. This information will be used to design travel routes, lift heights and slope configurations.

Waste Characterization Studies

A comprehensive geochemical characterization of all mine waste materials has been completed to support the development of an integrated water and waste management plan for the site. In developing the mine model, waste and host rock materials have undergone a comprehensive geological classification to ascertain the total volumes of materials that will be generated. Representative samples from each type of waste material were selected and tested for their acid generating and metal leaching potential as per the relevant guidance documents. The results of this study are reported under separate cover.



SRK Consulting Engineers and Scientists	Black Fox, Timmins, Ontario, Canada	**Black Fox Location Map**
SRK Job No.: 144418	Source: Mine Development Associates	
File Name: Figure 2-1.doc		Date: 07-10-07 · Approved: DKB · Figure: 2-1



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SRK Job No.: 144418

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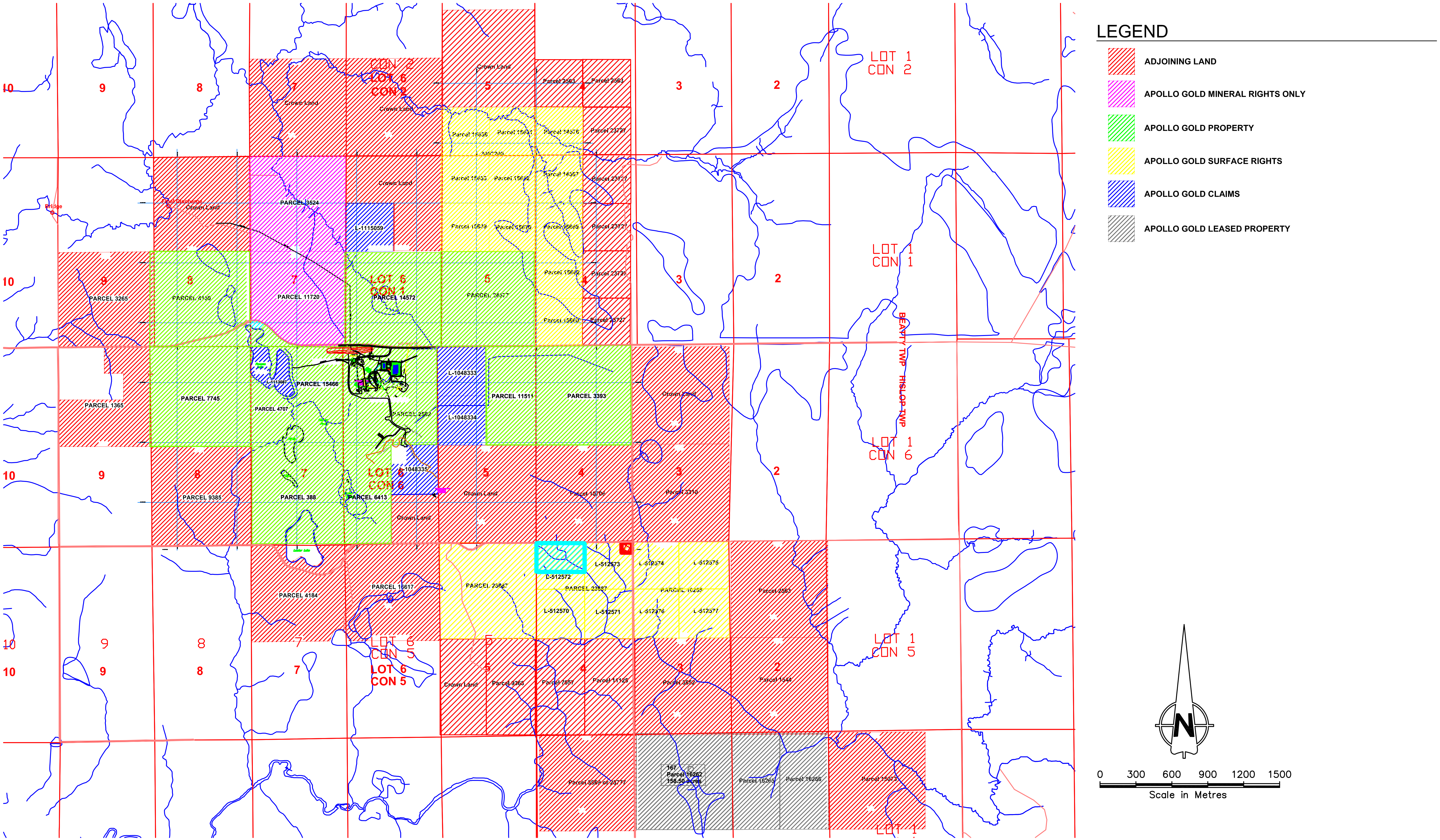
Black Fox, Timmins, Ontario, Canada

Source: Mine Development Associates

Adjacent Mines Location Map

Date: 07-10-07 | Approved: DBY | Figure: 2-2



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ApolloGold

BLACK FOX

CLAIM MAP

SRK JOB NO.: 0144418.00

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TIMMINS, ONTARIO, CANADA
SOURCE: APOLLO GOLD, INC.

DATE:	APPROVED:	FIGURE:	REVISION NO.
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3 Accessibility, Climate, Local Resources, Infrastructure and Physiography (Item 7)

3.1 Access

Black Fox is located 10km east of Matheson, Ontario and 65km east of Timmins, Ontario, Canada. Access is via Highway 101 East, which crosses the Black Fox claim block at the properties center from east to west. The mine site and its facilities are located on the south side of Highway 101 East. Supplies and services are available in Matheson or Timmins and can be delivered with a 12-hour turnaround. The primary industries are forestry and mining, and Black Fox is located in a well-established mining camp. Because of this, mining and exploration personnel as well as equipment can be found locally for projects in the area.

3.2 Climate

Temperature ranges from 20°C to 33°C during the summer months and -30°C to 10°C during the cooler winter months of October to May. The average precipitation is 873.4mm/yr and ranges between 44.5mm in February to 100.1mm in July. Rapid melting of accumulated snowfall can produce local flooding on the property for short periods during the spring months. Average monthly wind speeds for the region are 11 to 15km/hr (Dyck, 2007). Past operations at the property have not been affected by weather. The surface at Black Fox is mainly agricultural land with secondary growth of poplar and willow shrub.

3.3 Physiography

The Black Fox property area is predominantly agricultural land with a mature willow shrub, poplar, black spruce, and white birch forest located to the south and eastern borders of the property. The region is characterized by outwash deposits from continental glaciation including raised beaches, flat clay pans and eskers. Relief includes rock knobs and ridges (Prenn, 2006; Dyck, 2007).

Surface waters include lakes, rivers, and their associated habitats. Lakes include Froome Lake located 0.7km west of the mine, Leach Lake located 1.4km northwest of the mine and Lawler Lake located 1.7km south. Two other lakes, Salve located 5.2km north and Nickel located 5.9km north, form the headwaters of the Salve Creek. Salve Lake is designated as a Forest Reserve and Recommended Conservation Reserve (Dyck, 2007).

The property is located on the Salve Creek and Pike River watersheds, which are both tributaries of the Black River. The Black River flows north into to the Abitibi River which in turn flows into the Moose River. The Moose River ultimately flows into James Bay (Dyck, 2007). The Black Fox property has low to moderate topography with elevation ranging from 295 to 330m above mean sea level (amsl) (Prenn, 2006).

3.4 Local Resources and Infrastructure

The infrastructure of the Black fox Project consists of Highway 101 East, which is adjacent to the project site and facilities. The existing surface site facilities, shown in Figure 3-1, consist of the following infrastructure:

- A trailer complex, complete with administration office, mine dry facilities, geological/engineering offices, and shower/washroom facilities;

- Site access roads;
- A septic tank and tile field;
- A pump house at the east side of Froome Lake, and associated 750m long pipeline for the taking of fresh water for showers/toilets and drilling purposes;
- A 13,500L fresh water tank, in an insulated metal clad shed (water tank house);
- A compressor station;
- A core log shack;
- A former surface maintenance shop;
- One 4,500L diesel storage tank;
- An approved mine water treatment system, consisting of a settling pond, a polishing pond, and a spillway pond (for emergency discharge purposes);
- An acid addition building (where sulfuric acid and ferric sulphate are added for pH adjustment and arsenic removal, respectively), associated with the mine water settling pond treatment system;
- A 2.6km long, 150mm dia. HDPE pipeline, to Salve Creek, associated with treated mine water effluent;
- A downcast fan, with mine air heater, and a 30,000L propane tank;
- A main hydropower line to the on-site 5,000kVA transformer substation and distribution lines;
- A mine portal; and
- Waste rock and ore storage pad areas.

The existing infrastructure at the site will be removed and new infrastructure will be constructed to facilitate the mine development. New infrastructure to support open pit and underground mine operations will include buildings and structures, truck shop, laboratory, administration building, the firewater / freshwater pump house, and ancillary buildings. Future facilities are shown in Figure 3-2.

The Feasibility Study considered the option of including an on-site mill facility at Black Fox. However, this option was not selected since Apollo's acquisition of the Stock Mill from St. Andrew's. While the mill building and facilities are shown in Figure 3-2, it is not included in this analysis.

3.4.1 Buildings and Structures

Truck Shop

The Truck Shop will be a two-story, steel-frame, prefabricated, slab-on-grade, metal clad and insulated building measuring approximately 33m x 80m. This building will house offices, a first aid station, a training room, lunch room, men's and women's locker room, small vehicles repair shop, large vehicle repair bays, a truck wash station, and the warehouse. The building is located east of the mill building at the end of the main access road.

Laboratory

The laboratory facility will be a single-story, steel-frame, prefabricated, slab-on-grade building measuring approximately 18.3m x 42.7m. This building will house offices, a sample preparation area, sample storage, weighing room, wet lab, metallurgical lab, building mechanical services (including fume collection and dust collection), and washrooms. The building is located east of the truck shop on the main access road adjacent to the administration building.

Administration Building

The administration facility is a single-story, steel-frame, prefabricated, slab-on-grade building measuring approximately 18.3m x 42.7m. The building will provide offices, conference rooms, archiving, building mechanical services, and washrooms. The building will be located east of the process plant on the main access road.

Firewater/Freshwater Pump House

The Firewater/Pump house will be a single-story, steel-frame, prefabricated, slab-on-grade building measuring approximately 12m x 12.4m. The freshwater pump motor control center (MCC) will be located on the ground floor of the building next to the transformer. Room has been allocated to accommodate a potable water treatment package in the event that future regulations require such treatment.

Ancillary Buildings and Facilities

Prefabricated metal buildings (laboratory and administration buildings) have been quoted from preliminary layout drawings. The quotes did not include doors, windows, HVAC, etc., and these have been added as allowances.

3.4.2 Power Supply

The plant will be fed from an existing 27kV power line to the plant site. Power will be distributed at the plant site from this 27kV power line, a 5MVA, 4,160V, 3 phase, 60Hz distribution transformer will be installed near the milling area. This transformer will distribute power for the milling and crushing areas at 4160 volts. Maximum peak power demands for this plant are estimated at 6.2MW. A 750kW diesel-electric generator will provide emergency service. It was determined that the existing infrastructure will not be adequate to provide power for the plant, and all other loads at the mine site, so this current design will upgrade this system to adequately handle the new loads.

3.4.3 Water Supply and Distribution

The fresh water supply will be provided from the existing system to a fresh/fire water storage tank. Fresh water will be supplied to the insulated fresh water/fire water storage tank. One electric and one diesel-powered pump will afford fire protection.

About 25m^3/hr of make up water is required for the mill operation, with the majority of water for the operation coming from recirculation of water from the tailings facility.

The underground is currently being pumped at the rate of 25 to 35m^3/hr. Mine run off is anticipated to average above 16m^3/hr. These two sources will report to the tailings impoundment and provide the volume required for mill make up water.

Excess water will be treated during the spring and summer month for discharge to the environment through the water treatment system.

3.4.4 Fire Protection

The Fresh /Firewater tank will provide both fresh water and firewater. Firewater will be fed by an electric firewater pump with a diesel backup pump in the event of a power failure. The firewater pump will deliver firewater through underground piping to the hydrants and firewater stations in the plant site and ancillary buildings area. The fresh water discharge connection is at an elevation above the tank bottom and ensures the remaining volume will be available for firewater purposes. Distribution will consist of a buried ring main around major facility buildings with hydrants and stand pipes connected to indoor hose stations. Municipal fire department is located within 15km of the Black fox Project.

3.4.5 Access Roads

The Black Fox Project is located 10km east of Matheson, Ontario, Canada on Highway 101, which runs through the middle of the property north of the ore body, the process facility, and the ancillary buildings. The property is also contacted by two other roads: Hislop 2 Road to the east and Hislop 6 Conc to the south. The site access road currently approached the administration building from highway 101. The new main access road will be approximately 1.5km long oriented in an east-west direction intersecting Hislop 2 Road approximately 0.8km south of highway 101.

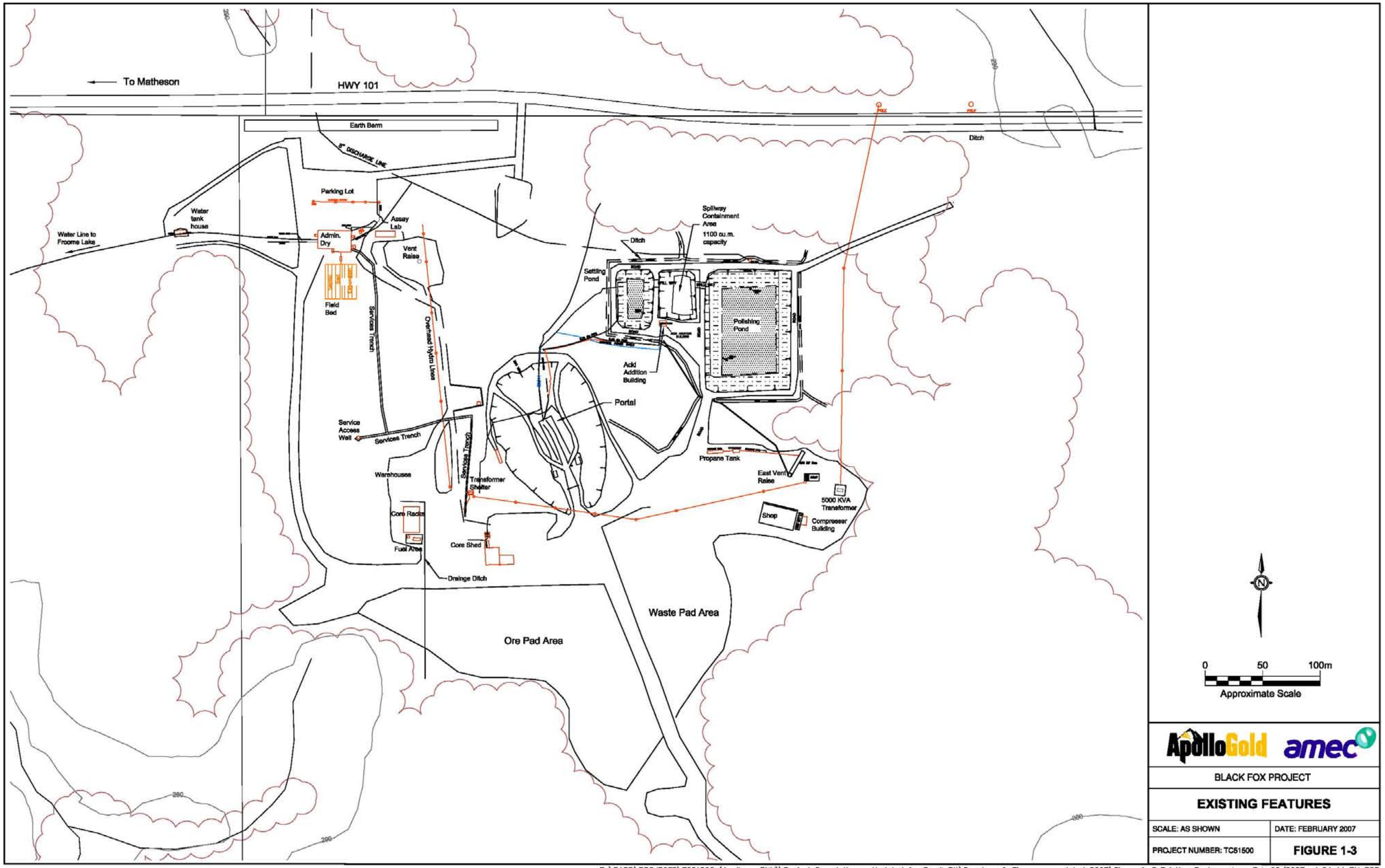
The design criteria for the main access road are based on a number of factors, such as design speed, vehicle types, etc., establishing the minimum geometric design elements for a road. These elements include vertical and horizontal alignment and sighting distances. Other design aspects are surface requirements and shoulder widths, horizontal clearances, etc. that are not directly related to design speed. Since this is not a large-scale project and traffic volumes are relatively low, a gravel road was selected, which is typical for this size of project in Canada.

3.4.6 Security

A total of 6,388m of fencing will be installed to limit access to the plant site, the pit and tailings impoundments. A security gate and guardhouse will be positioned on the main access road at the entry point to the project area. Road entry will controlled by a guardhouse operated 24 hours per day, 365 days per year.

An experienced Canadian security services company will be contracted to handle all site security; the contractor will report to the Safety and Security Supervisor, who reports to the Environmental, Health and Safety Manager. The contracted security team will include a supervisor and two guards per shift, and the company will supply its own vehicles and equipment. The security team’s responsibilities will include maintaining a constant, 24/7 presence at the site access guardhouse, performing roving patrols around the site, and performing plant security and loss protection.

Refined gold ore materials will be secured in a controlled access locked reinforced concrete and concrete block building located within the mill processing facility.



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SRK Consulting Engineers and Scientists	Black Fox, Timmins, Ontario, Canada	Existing Infrastructure
SRK Job No.: 144418		
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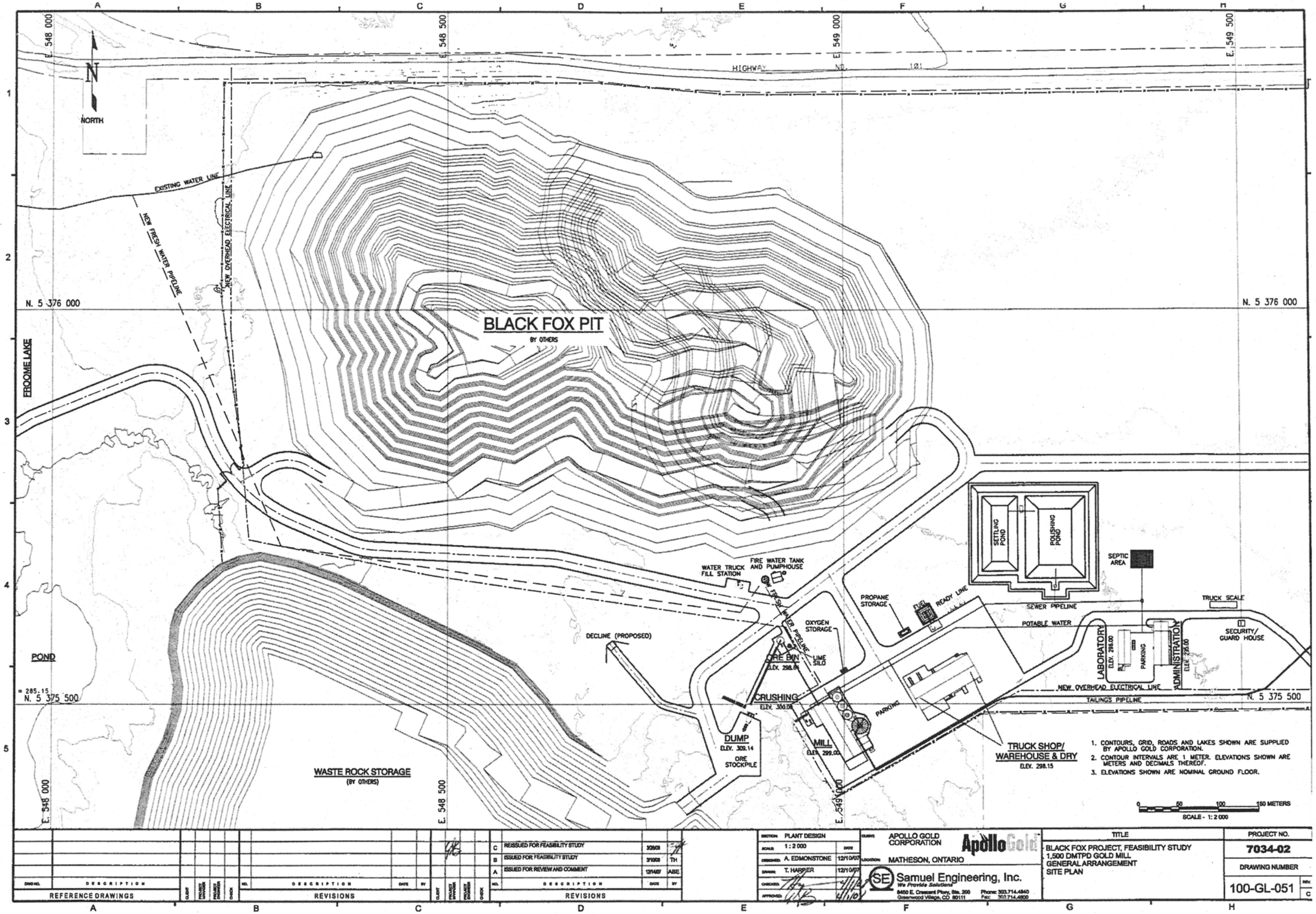

SRK Consulting Engineers and Scientists 7175 West Jefferson Ave. Suite 3000 Denver, Colorado 80235 303-985-1333	ApolloGold	BLACK FOX GENERAL ARRANGEMENT SITE PLAN
SRK JOB NO.: 0144418.00 / TASK 0005 FILE NAME: 014418.00.Fig.3-2.General.Arrange.Site.Plan.dwg	TIMMINS, ONTARIO, CANADA SOURCE:SAMUEL ENGINEERING, INC.	DATE: MARCH 2008 · APPROVED: NM · FIGURE: 3-2 · REVISION NO. A

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4 History (Item 8)

4.1 Ownership History

The property was first explored by Dominion Gulf in 1952 and then by Hollinger in 1962. In 1988, Glimmer Mine Inc. put together the property package using a combination of crown and private lands. In 1989, Noranda Exploration Company Ltd. (Noranda) entered into a joint venture agreement with Glimmer. As a result of this agreement, Noranda held a 60% interest in the property. During their ownership, Noranda merged with Hemlo Gold Mines Inc. (Hemlo). Exall purchased the property from Hemlo in April 1996, obtaining approximately 60% interest in the property with Glimmer retaining 40%. Apollo acquired a 100% ownership in the fall of 2002 and renamed the property “Black Fox” (Prenn, 2006).

4.2 Exploration History

4.2.1 Drilling

The first drilling on the property was done by Dominion Gulf in 1952. Hollinger next tested the area in 1962 near the diabase dikes located in the easternmost part of the property. Between 1989 and 1994, Noranda, and later Hemlo, completed eight surface diamond drill programs with a total of 27,800m of drilling in 142 drillholes. The result of these drilling programs was the definition of an intensive grouping of ore zones in two areas of the property. These ore zones were all within 250m of the surface. Some high-grade intercepts, including abundant visible gold, were recovered during the drilling program. Between 1995 and 1999, Exall completed another 142 surface diamond drillholes, as well as 720 underground diamond drillholes with mine development (Dyck, 2007).

4.2.2 Mapping and Geophysics

Noranda first performed detailed geological mapping of the property and much of the surrounding area in 1989. This data has provided a very good base of information from which subsequent workers have determine structural trends and location of the most favorable stratigraphic units (Dyck, 2007).

The property has had a number of different geophysical surveys completed by different previous owners in combination with various drilling programs. In conjunction with Noranda’s 1989 drilling program, a total field magnetic survey over most of the property was conducted by Exsics Exploration Ltd. Noranda also had Lamontagne Geophysics Ltd. complete an Inductive Source Resistivity survey and R.S. Middleton Exploration Services conduct a conventional IP survey over portions of the property at that time Additional IP surveys were completed in 1997 for Glimmer by JVX Ltd. This later survey was limited to the area adjacent to the mine (Dyck, 2007).

Exploration was also conducted using geological, magnetic and gradiometer surveys conducted by the University of Toronto Electro-Magnetometer (UTEM) survey, and a limited induced polarity (IP) survey (Prenn, 2006).

The highly magnetic anomalies have assisted in the mapping of the basalt and ultramafic units on the property. In addition to this, low magnetic trends may be indicative of hydrothermal alteration that altered the magnetic qualities of the surrounding rocks (Dyck, 2007).

4.3 Historic Resource and Reserve Estimates

The historic resource estimates are summarized in Table 4.3.1. All of the resource estimates include reserves, except the 1998 estimate which is resource only and does not include the 1998 reserves listed in Table 4.3.1. Table 4.3.2 summarizes the historic reserve estimates made on the Black Fox deposit. The historical reserve and resource estimates performed before the MDA 2006 estimates, pre-dates the development of NI 43-101 reporting guidelines and was not estimated in compliance with NI 43-101 procedures.

Table 4.3.1: Historic Resource Estimates

Year	Measured			Indicated			Total Measured and Indicated			Inferred			Estimator
	kt	Grade gpt-Au	koz-Au	kt	Grade gpt-Au	koz-Au	kt	Grade gpt-Au	koz-Au	kt	Grade gpt-Au	koz-Au	
1994							727	11.30	264				Hemlo (Jarvi)
1996							551	11.52	204				Roscoe Postle
1996							678	11.30	246				Roscoe Postle
1998	44	4.84	7	154	5.58	28	198	5.42	34	382	10.33	127	Exall
1999	410	7.27	96	796	8.20	210	1,205	7.88	306	274	5.96	52	Exall
2000	586	6.93	131	1,022	7.36	242	1,608	7.20	372	381	6.65	81	Exall
2001	268	4.09	35	566	4.93	90	833	4.66	125	353	7.00	79	Exall
2006										7,854	4.89	1.2	MDA

Note: All resources include material reported as reserves except 1998 which is in addition to reserves, Roscoe Postle audited all Exall Estimates

Table 4.3.2: Historic Reserve Estimates*

Year	Proven			Probable			Total Proven & Probable			Estimator
	kt	Grade gpt-Au	koz-Au	kt	Grade gpt-Au	koz-Au	kt	Grade gpt-Au	koz-Au	
1996				499	11.14	179	499	11.14	179	Canadian Mine Development (Feasibility)
1996				477	10.70	164	477	10.70	164	Bharti Engineering Associates
1996				621	11.60	232	621	11.60	232	Roscoe Postle
1997				665	12.90	275	665	12.90	275	Roscoe Postle
1998	330	9.88	105	488	10.32	162	818	10.14	267	Exall
1999	284	8.48	77	553	9.50	169	837	9.15	246	Exall
2000	422	7.82	106	560	8.93	161	981	8.45	267	Exall
2001	303	8.45	82	475	9.21	141	778	8.92	223	Exall
2006				3,063	4.56	449	3,063	4.56	449	MDA

*All resource estimates prior to MDA in 2006 are historical and were not reported to NI 43-101 compliance.

4.4 Production History

Ore mined from Black Fox was custom milled from 1997 through September 1999 at the St. Andrew Goldfields Stock Mill located 34km from the mine. From October 1999 through May 2001, ore was milled at Kinross Gold's Macassa facility in Kirkland Lake, subsequent to mineral tests carried out by Lakefield Research and other metallurgical laboratories. These mills used cyanidation of the whole ore to process the ore. Testwork has indicated that gravity pre-concentration may improve gold recovery (Prenn, 2006).

Black Fox was formally owned and operated by Exall. The previously estimated ore reserves were 3.1Mt with a grade of 4.6gpt-Au (449koz-Au) all from open pit mining (Prenn, 2006). The open pit total waste is 47.2Mt of waste rock and overburden material with an equivalent overall strip ratio of 15.4 waste: 1 ore. The underground ore resources (below 9,815m) were 1.6Mt with a grade of 8.1gpt-Au.

Table 4.4.1 summarizes the reported gold production of 210.8koz from the Black Fox property, with the grades required at 100% recovery. Figure 4-1 illustrates several views of the underground workings of the mine at end of year 2000.

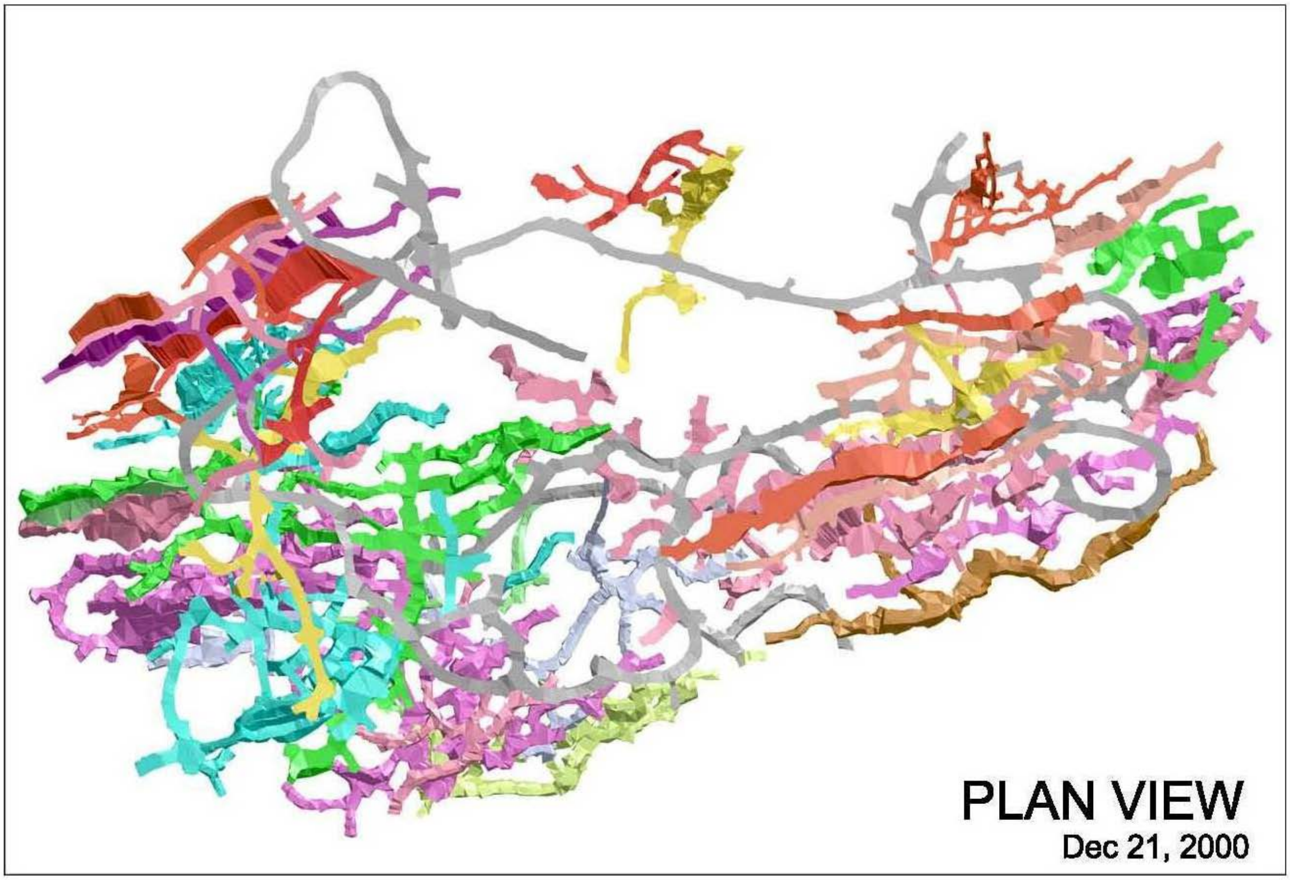
Table 4.4.1: Black Fox Project Production History*

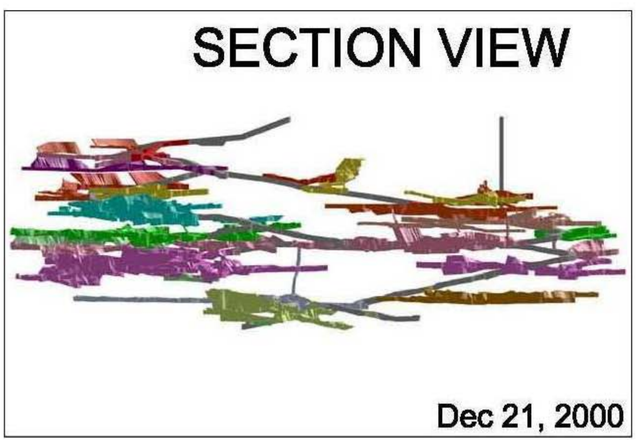
Year	kt	Grade gpt-Au	koz-Au
1997	194	6.79	40
1998	309	6.67	64
1999	259	5.82	48
2000	255	5.82	46
2001	82	4.81	12
Total	**1,099**	**5.97**	**211**

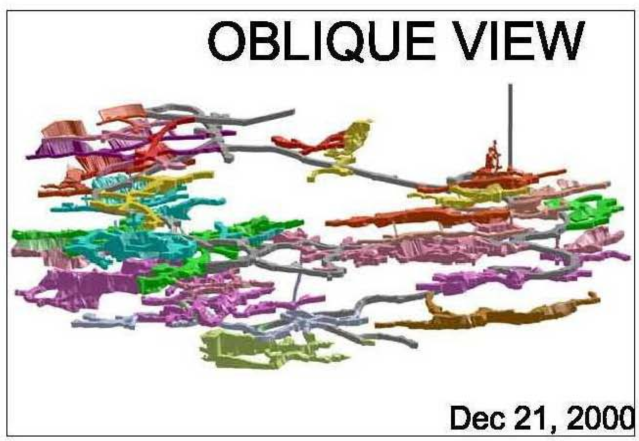
*Actual reported production.

Exall mined portions of the deposit from the bottom of the crown pillar to the 225m level (measured vertically 225m below the surface) using conventional underground mining methods including jumbo drills, diesel load haul dump (LHD's) loaders and haul trucks in a random room and pillar method. The limited amount of surface or underground core drilling that was completed by Exall did not allow for detailed mine planning, subsequently the daily mining production planning was determined by management and geological decisions at the face before each round was mined as ore or waste.

Comparing the reserves estimated in Table 4.3.2 to historic production in Table 4.4.1 shows that the grade and tonnage estimates are not very close to the actual production of about 1.1Mt with and average grade of approximately 6gpt-Au. The estimates between 1996 and 1997 show a range from 162koz to 275koz-Au. In 2001 the reserve estimate was 140koz, most of which is still in the ground. All of the historic reserve estimates show higher grades and less tonnes than were actually mined during historic production (Prenn, 2006).







SRK Consulting Engineers and Scientists	**Black Fox, Timmins, Ontario, Canada**	**Black Fox Underground Workings**
SRK Job No.: 144418	**Source: Mine Development Associates**	
File Name: Figure 4-1.doc		Date: 07-10-07 | Approved: DKY | Figure: 4-1

5 Geological Setting (Item 9)

5.1 Regional Geology

The Black Fox deposit is east of the city of Timmins in northeastern Ontario located on the Destor Porcupine Fault Zone (DPFZ). The DPFZ has a strike length of about 200km, and many of Ontario's gold mines are located on or near the DPFZ.

5.1.1 Lithology

The Black Fox property is located within Precambrian age metavolcanics and metasedimentary rocks of the Abitibi Greenstone Belt. This is one of the world's largest Archean greenstone belts believed to have formed by a complex history of paired arc volcanism and back arc sediments subsequently deformed during continental collision. The area hosts five main rock groups, most of which have tectonic contacts of varying intensity. These include:

- Blake River Group;
- Kinojevis Group;
- Stoughton-Roquemaure Group (Black Fox Host Units);
- Hunter Mine Group; and
- Porcupine Group.

The Blake River Group consists of calc-alkalic basalt, andesite, dacite and rhyolite flows and tuffs. It is the youngest of the volcano-sedimentary rocks and stratigraphically overlies the Kinojevis Group. The Kinojevis Group is a sequence of iron rich tholeiitic volcanic rocks that occur on both sides of the Blake River synclinorium. The Stoughton-Roquemaure Group stratigraphically underlies the Kinojevis Group and is a mixture of ultramafic to basaltic komatiite lavas and Mg-rich tholeiitic basalts that host the Black Fox gold zones. This is underlain by calc-alkalic rocks of the Hunter Mine Group. The Hunter Mine Group consists primarily of calc-alkalic pyroclastic and flow rocks in the dacite-rhyolite compositional range. The Porcupine Group of wacke, siltstone and argillite sediments are the youngest in the region. They are separated from the above mentioned volcanic groups by a major fault contact interpreted to have once been a thrust fault. This group lies predominantly north of the Black Fox property. Pre- to syn-kinematic granitic rocks occur throughout the section, cross-cutting all older lithologic units. (Hoxha and James, 2007). The tectono-stratigraphic column of the Black Fox area is shown in Figure 5-1.

5.1.2 Structure

The Black Fox property is situated within a deeply rooted ductile shear zone accompanied by large-scale isoclinal folds. The mineralization is situated on the southern limb of a regional anticline and on the northern limb of the Blake River Syncline. At Black Fox, the axial plane of the syncline strikes roughly NW-SE. The Black Fox deposit is located within the DPFZ. It was first recognized in the early 1900's with the discovery of gold deposits in the Timmins area. The DPFZ extends for over 200km, from Timmins in the west, to the Duparquet area of Quebec to the east and hosts many of Canada's richest gold mines. The DPFZ hosts gold mineralization comparable to the Cadillac, to the South and Casa Berardi Fault Zones located to the North. These regional fault fabrics typically strike east to southeast and dip to the south. They are deeply rooted structures that likely penetrate to the mantle, as indicated by the associated ultramafics of the DPFZ and the syenites of the Ross Mine Syenitic Belt (RMSB). Zones of intense hydrothermal alteration measured in thousands of feet are locally associated with these belts. These types of deep-rooted faults are considered to be the main channel way for the upward migration of deep fluids. The main structural feature on the Black Fox property is the intersection of the DPFZ with the RMSB (Hoxha and James, 2007). Figure 5-2 illustrates the regional geology of the area.

5.2 Local Geology

Most of the Black Fox area is rather flat and lacking in outcrops. Pleistocene overburden averages 20m thick and is composed of lacustrine clay, gravel and till. The main bedrock types consist of variably sheared, faulted, carbonatized and mineralized sequences of komatiitic ultramafic volcanics, belonging to the Stoughton-Roquemaure Group. These strike northwest-southeast across the property, dipping 45^{o} southwest, parallel to the DPFZ. The komatiites are strongly altered to a bleached, light grey-buff color with pervasive ankerite-talc and ankerite-quartz-sericite-fuchsite assemblages. This alteration package is underlain to the north by a thin, fine grained, green greywacke metasedimentary unit, a thick sequence of massive to pillowed tholeiitic mafic volcanic rocks and by the regionally extensive package of argillites and wackes of the Porcupine Group sediments (Hoxha and James, 2007).

To the south, and forming the hanging wall of the main ankerite zone are relatively undeformed very fine-grained, green pillowed tholeiitic mafic volcanics with lesser intercalated black chlorite-serpentine, chlorite and talc-chlorite altered komatiitic ultramafic flows (Hoxha and James, 2007).

Numerous syenitic and feldspar ± quartz porphyry sills and dykes of various ages occur, primarily within the main ankerite alteration zone. They are commonly massive to brecciated, silicified and pyritic with occasional sericite and hematite alteration and a more common black chlorite alteration at the contacts. They vary in color from pink, grey, whitish, yellow, pale green and reddish. Fragments of these dykes frequently occur within the more strongly deformed green carbonate zones and they can contain very high gold grades (Hoxha and James, 2007).

Very narrow to massive, dark green to buff-green mafic dykes and sills commonly occur within the main ankerite zone. They are generally weakly altered and probably post-date much of the alteration and deformation. Diabase dykes are the youngest rocks in the area, occupying very late north-striking crustal fractures. Figure 5-3 illustrates the local geology, and Figures 5-4 through 5-6 illustrate typical cross sections through the deposit (Hoxha and James, 2007).

5.3 Mine Geology

Surface, underground and exploration drilling has delineated five major rock types in the vicinity of the Black Fox mineralization. These include:

- Mafic volcanic units;
- Metasediments;
- Green carbonate schist;
- Ultramafic volcanics; and

- Felsic intrusive units.

5.3.1 Mafic Volcanic Units

The mafic volcanic units are further subdivided into massive mafic volcanics (MV), pillowed mafic volcanics (PMV) and bleached mafic volcanic flows (BMV). The MV and PMV are fine grained typically hosting a significant degree of chlorite alteration. These units occur primarily within the hanging wall of the deposit. In the hanging wall, they are fractured and contain minor amounts of quartz-calcite veins. Where they occur in the footwall, they lack carbonate veining and have more prevalent quartz and chlorite alteration (Hoxha and James, 2007).

The BMV, also known as the “Flow Zones”, is a medium to fined grained, bleached mafic volcanic rock which is generally located just above the footwall of the mineralization. This unit has weak chlorite and sericite alteration and is associated with fine grained disseminated pyrite. Stronger sericite and pyrite alteration is found near the upper contact of the BMV. Pyrite in this unit is associated with gold. The BMV dips 45 to 55° SW and is moderately foliated. Two quartz vein arrays have been recognized within this unit. The first is a pre-tectonic vein set parallel to the foliation and the second is a series of late veins perpendicular to the foliation (Hoxha and James, 1998).

5.3.2 Metasediments

The metasedimentary rocks overlie the BMV and also occur as lens of greywacke (SED) within the green carbonate schists (CGR) as described below. At the top of the BMV, the greywacke layers are interbedded with siltstone. This unit is discontinuous, varies from 0.05m to 1m thick and displays graded bedding with stratigraphic tops to the southwest. Its color ranges from pale green to yellowish where well developed sericite alteration is present. This alteration typically extends over 1 to 2m wide zone and can be associated with gold mineralization (Hoxha and James, 2007). Greywacke lens occurring within the CGR are yellowish with strong sericite alteration. Generally, they are less than 2m thick, ranging up to 4m (Hoxha and James, 2007).

5.3.3 Green Carbonate Schist

The CGR, ranges from 15m to 75m thick and is continuous along strike and dip across the property. It is characterized by intense ductile and brittle deformation shown by multiple generations of foliation and veining. The CGR host a quartz-ankerite-fuchsite-leucoxene alteration assemblage accompanied by varying levels of retrograde chlorite alteration. This unit contains numerous small bodies and blocks of felsic dikes and sills with a syenitic composition. A complex stockwork of quartz-ankerite veins cross cut the main CGR fuchsite assemblage and the felsic material. This stockwork is accompanied by intense hydrothermal alteration. Locally, grey carbonate fragments, from lapilli sized (~2mm) to 1 to 2m angular blocks are found in the CGR. The lapilli sized fragments have been deformed to their current elliptical shape, elongate parallel to foliation. Mineralogy, microscopic texture and structures suggest that the CGR is an ultramafic pyroclastic rock which has undergone intense ductile deformation. Medium to coarse-grained pyrite is a minor component and is estimated at approximately 1%. Gold occurs as fine-grained free gold located along chlorite slips, as disseminated grains in quartz veins and associated with the felsic dikes (Hoxha and James, 1998; 2007).

5.3.4 Ultramafic Volcanics

The ultramafic volcanic rocks are divided into five units. These include; chlorite-talc ultramafic (CUV), talc ultramafic (TUV), grey carbonate (CGY), silicified grey carbonate (SUV) and ankerite ultramafic (AUV). Generally, the ultramafic volcanics occur stratigraphically above the CGR.

The CUV is dark green, massive, brecciated in places and often magnetic ultramafic rock. This unit does not display pervasive carbonate alteration and carbonate is restricted to late veins and fractures. Tremolite is present, the two primary mineral assemblages are tremolite-talc-chlorite and talc-chlorite-carbonate. Locally, the CUV occurs within the mineralized envelope as a non-brecciated unit and the CUV is not of major economic significance (Hoxha and James, 1998; 2007).

The TUV is pale green-grey, fine grained, marbled with quartz-ankerite fragments and massive ultramafic volcanic rock. It tends to be strongly foliated proximal to shear zones, ranging from 0.3m to 15m thick. It is most often associated with the stockwork CGY (Hoxha and James, 1998; 2007).

The CGY is composed of a fine grained, massive matrix composed primarily of magnesite-quartz. Relic outlines of pyroxene and preserved black chromite grains are visible in hand specimen. This unit contains several generations of quartz veining. The CGY is 0.5m to 2m thick, generally occurs above the CGR and is bound by talc ultramafic shear zones (Hoxha and James, 1998; 2007).

The intensity of silicification and amount of quartz stockwork is the distinguishing characteristic between the CGY and the SUV. The SUV is very similar in appearance to the CBY, but the SUV is harder due to silicification. Two types of carbonatization-silicification have been observed at Black Fox:

- Impregnation of the original ultramafic volcanic rock by CO_2 and silica-rich fluids throughout the network of micro-fractures and cavities/porosity; and
- Silicification of the altered ultramafic volcanic rock by silica-rich fluid circulating throughout rectilinear centimeter-wide extensional fractures associated with shear zones.

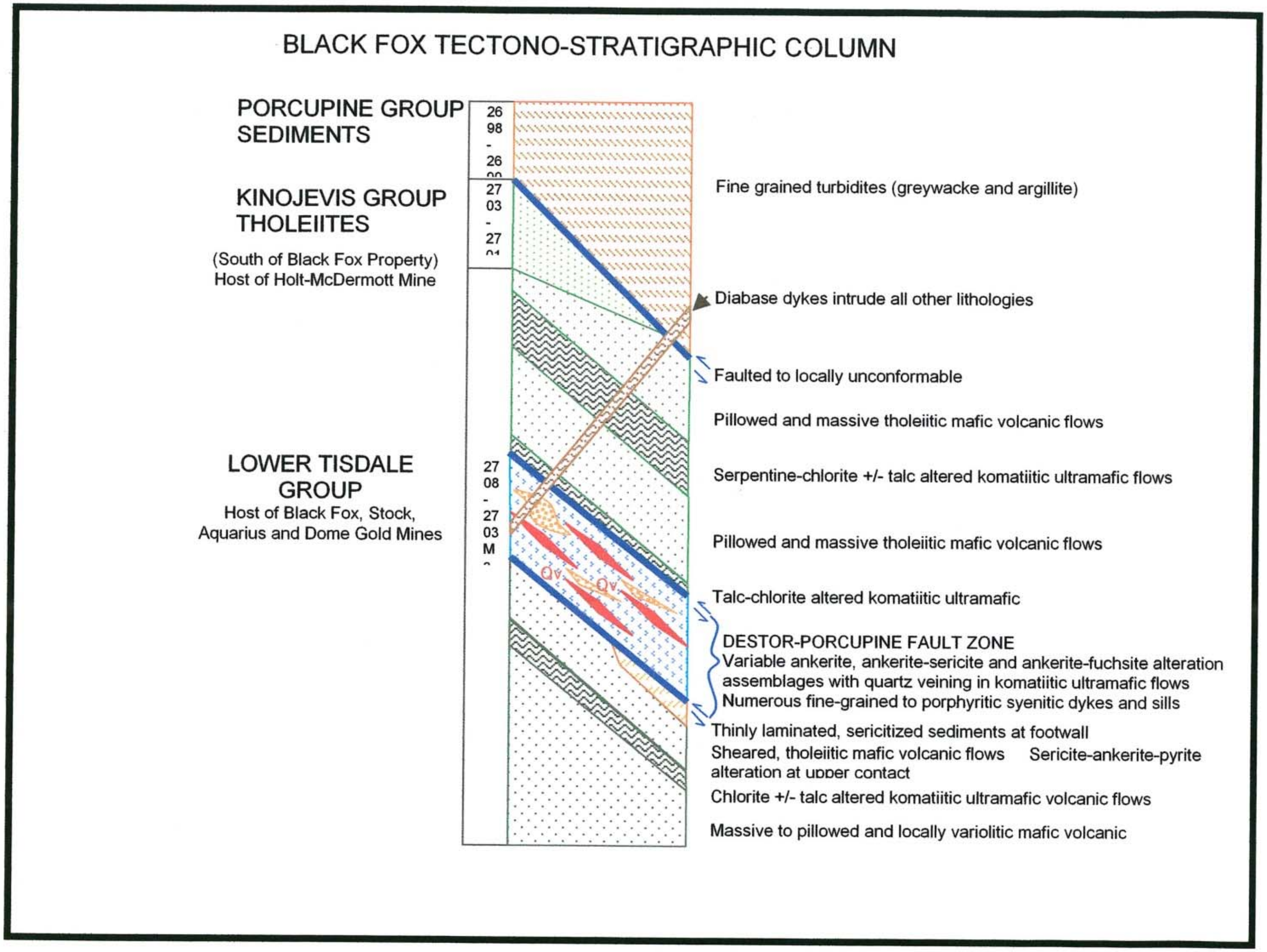
The CGY formed by the first process is moderately to strongly fractured, while CGY formed by the second process tends to be massive. Both the CGY and SUV host visible gold and are of economic importance at the Black Fox property (Hoxha and James, 1998; 2007).

The AUV is dark green-brown, fine–medium grained rock composed of a quartz-ankerite-calcite-chlorite assemblage cross cut by quartz-ankerite veining. Chloritization varies throughout this unit with matrix ankerite and calcite alternating downward through the package. Visible gold occurs in highly chloritized area as well as in association with the quartz-ankerite stockwork. The AUV generally occurs above the CGR and is one of the dominant rock types at the Black Fox property (Hoxha and James, 1998; 2007).

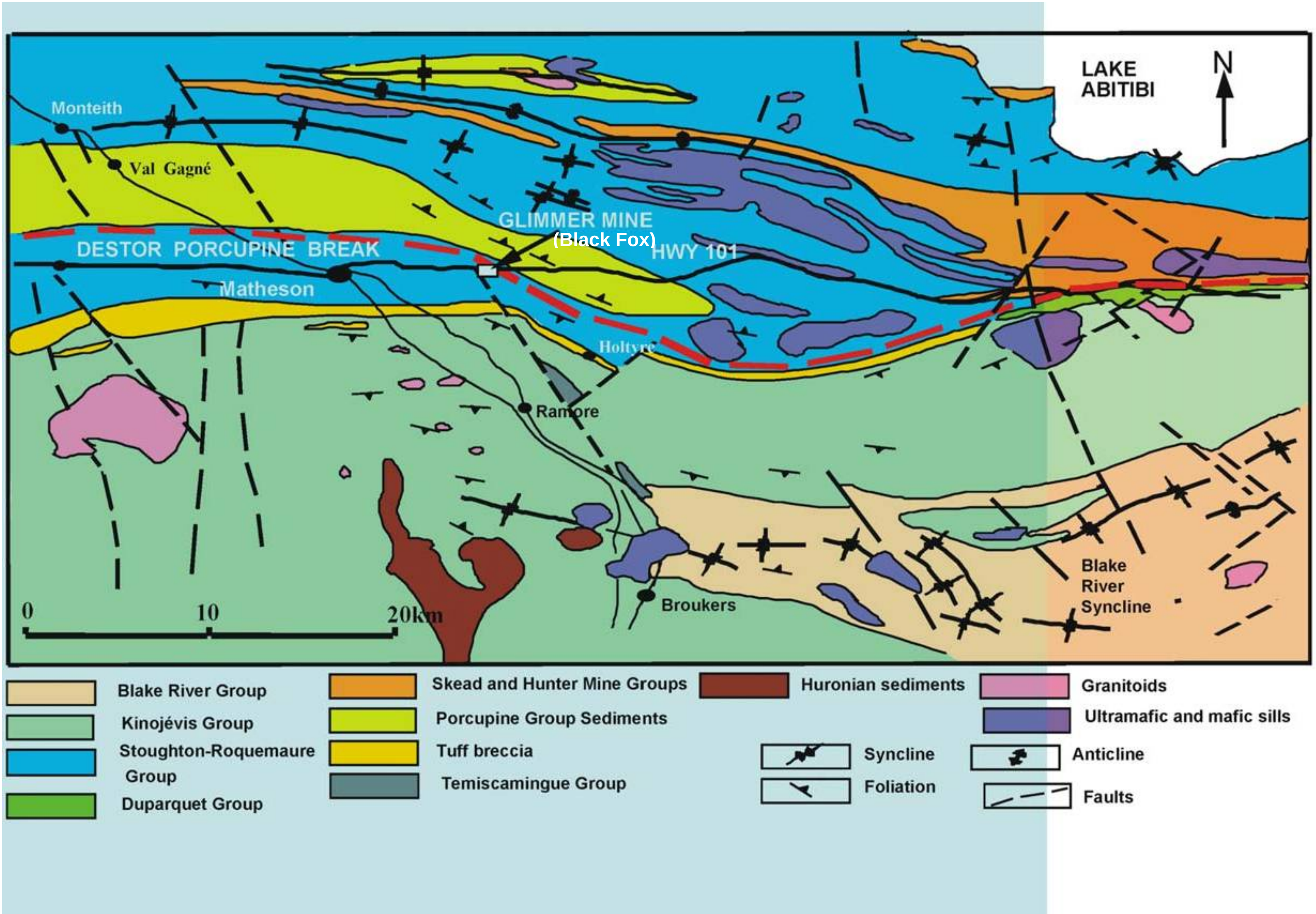
5.3.5 Felsic Intrusive Units

Many types of felsic intrusive (FI) have been recognized within a number of different lithologies at Black Fox. These range in color from grey to yellowish to reddish brown as a result of different alteration types. Most of the felsic intrusives are fine to medium grained, massive and moderately fractured, but some coarser grained porphyritic bodies have also been observed.

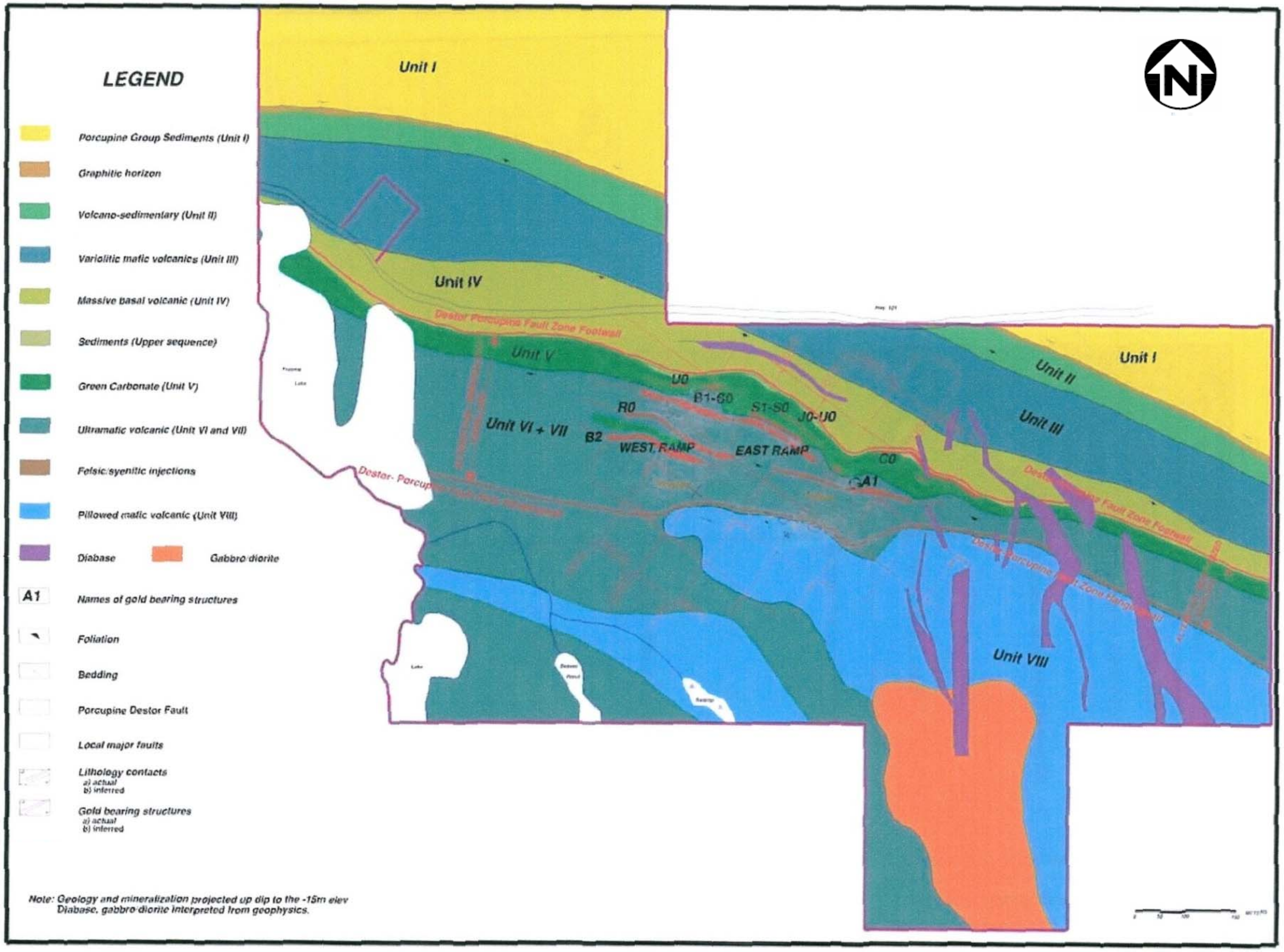
Generally, the felsic rocks are discontinuous, lensoidal in shape and aligned with the foliation of the host rock. They are often cross cut by quartz-ankerite stockwork and most are strongly affected by sericite and albite alteration. Varying amounts of fine-grained disseminated pyrite are a strong indication of gold mineralization. Gold occurs as free gold associated with quartz veins. Syenitic pods have been observed in the CGR. These are pink, coarse grained and contain a relatively high concentration of pyrite, at 5-15% they typically have an average gold grade of 15gpt (Hoxha and James, 1998; 2007).



SRK Consulting *Engineers and Scientists*	**Black Fox, Timmins, Ontario, Canada**	**Black Fox Tectono-Stratigraphic Column**
SRK Job No.: 144418		Date: 07-17-07 / Approved: DKY / Figure: 5-1
File Name: Figure 5-1.doc	**Source: Hoxha and James 2007**	



SRK Consulting Engineers and Scientists	Black Fox, Timmins, Ontario, Canada	Regional Geology
SRK Job No.: 144418	Source: Hoxha and James; 1998, 2007	
File Name: Figure 5-2.doc		Date: 07-17-07 Approved: DKB Figure: 5-2



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SRK Job No.: 144418

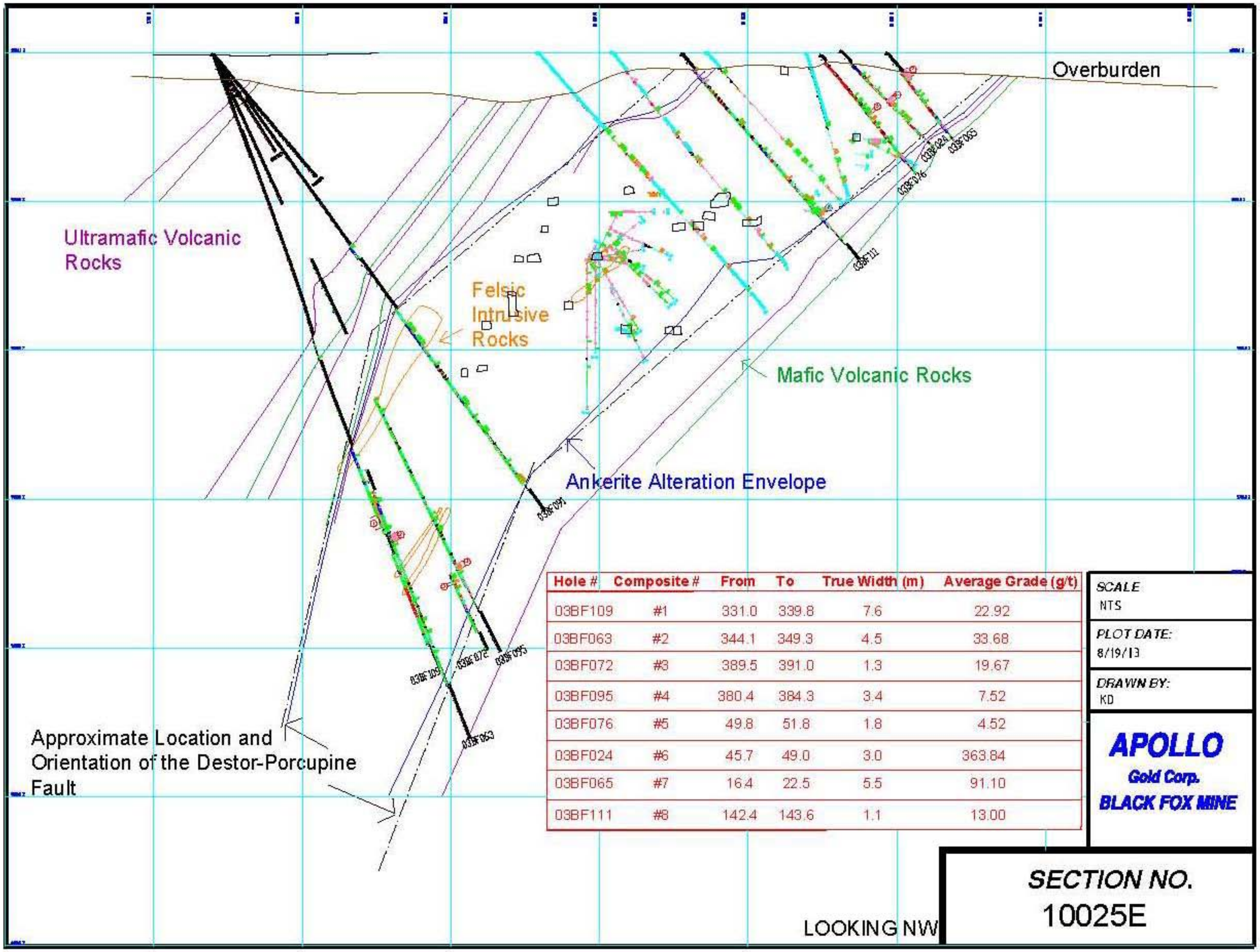
File Name: Figure 5-3.doc

Black Fox, Timmins, Ontario, Canada

Source: Hoxha and James; 1998, 2007

Local Geology

Date: 07-17-07 | Approved: DKB | Figure: 5-3



Hole #	Composite #	From	To	True Width (m)	Average Grade (g/t)
03BF109	#1	331.0	339.8	7.6	22.92
03BF063	#2	344.1	349.3	4.5	33.68
03BF072	#3	389.5	391.0	1.3	19.67
03BF095	#4	380.4	384.3	3.4	7.52
03BF076	#5	49.8	51.8	1.8	4.52
03BF024	#6	45.7	49.0	3.0	363.84
03BF065	#7	16.4	22.5	5.5	91.10
03BF111	#8	142.4	143.6	1.1	13.00

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SRK Job No.: 144418

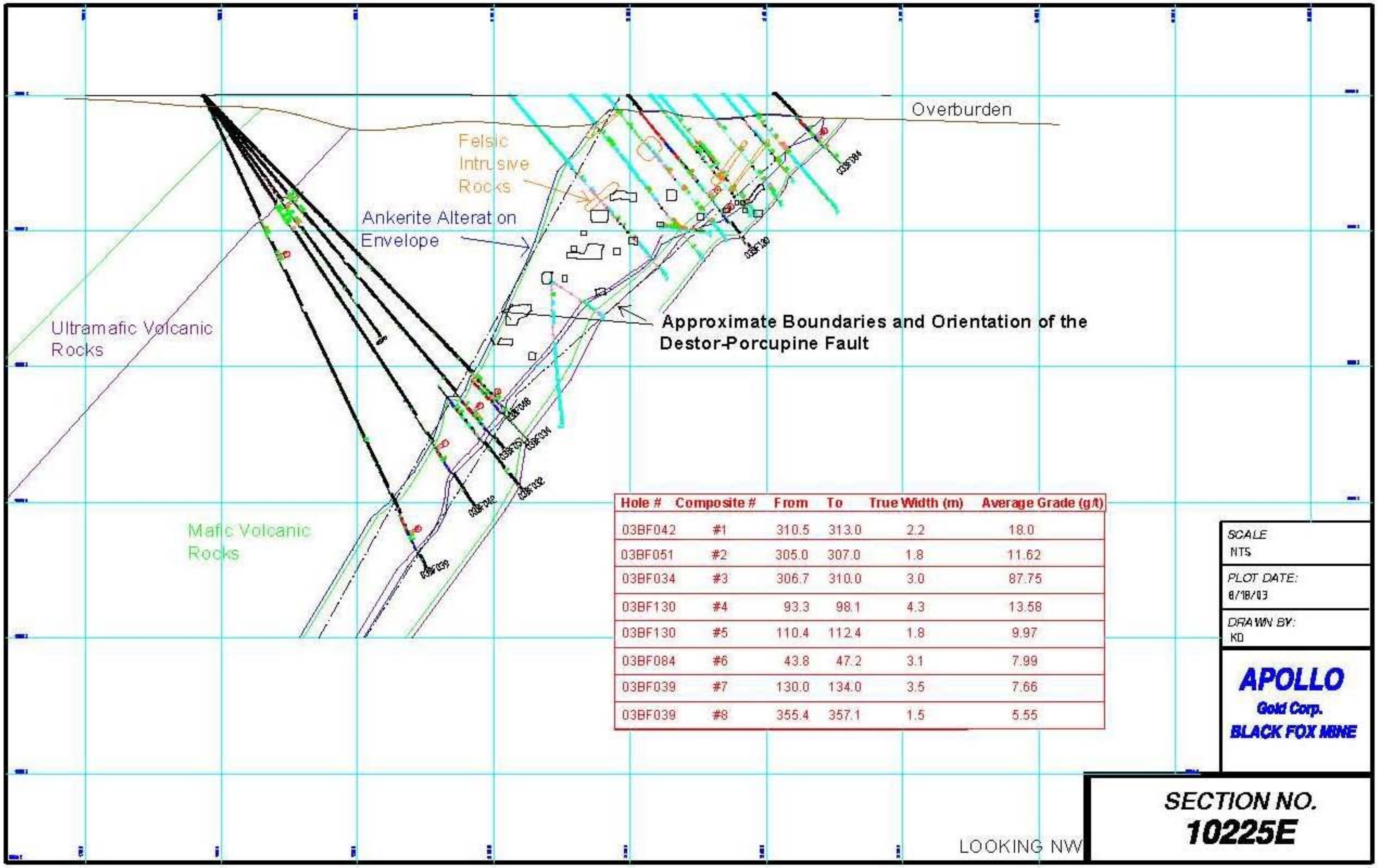
File Name: Figure 5-4.doc

Black Fox,
Timmins, Ontario, Canada

Source: Mine Development Association

Typical Cross Section 10025E

Date: 07-10-07 | Approved: DKY | Figure: 5-4



Hole #	Composite #	From	To	True Width (m)	Average Grade (g/t)
03BF042	#1	310.5	313.0	2.2	18.0
03BF051	#2	305.0	307.0	1.8	11.62
03BF034	#3	306.7	310.0	3.0	87.75
03BF130	#4	93.3	98.1	4.3	13.58
03BF130	#5	110.4	112.4	1.8	9.97
03BF084	#6	43.8	47.2	3.1	7.99
03BF039	#7	130.0	134.0	3.5	7.66
03BF039	#8	355.4	357.1	1.5	5.55

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SRK Job No.: 144418

File Name: Figure 5-5.doc

Black Fox,
Timmins, Ontario, Canada

Source: Mine Development Associates

Typical Cross Section 10225E

Date: 07-10-07 | Approved: DKY | Figure: 5-5

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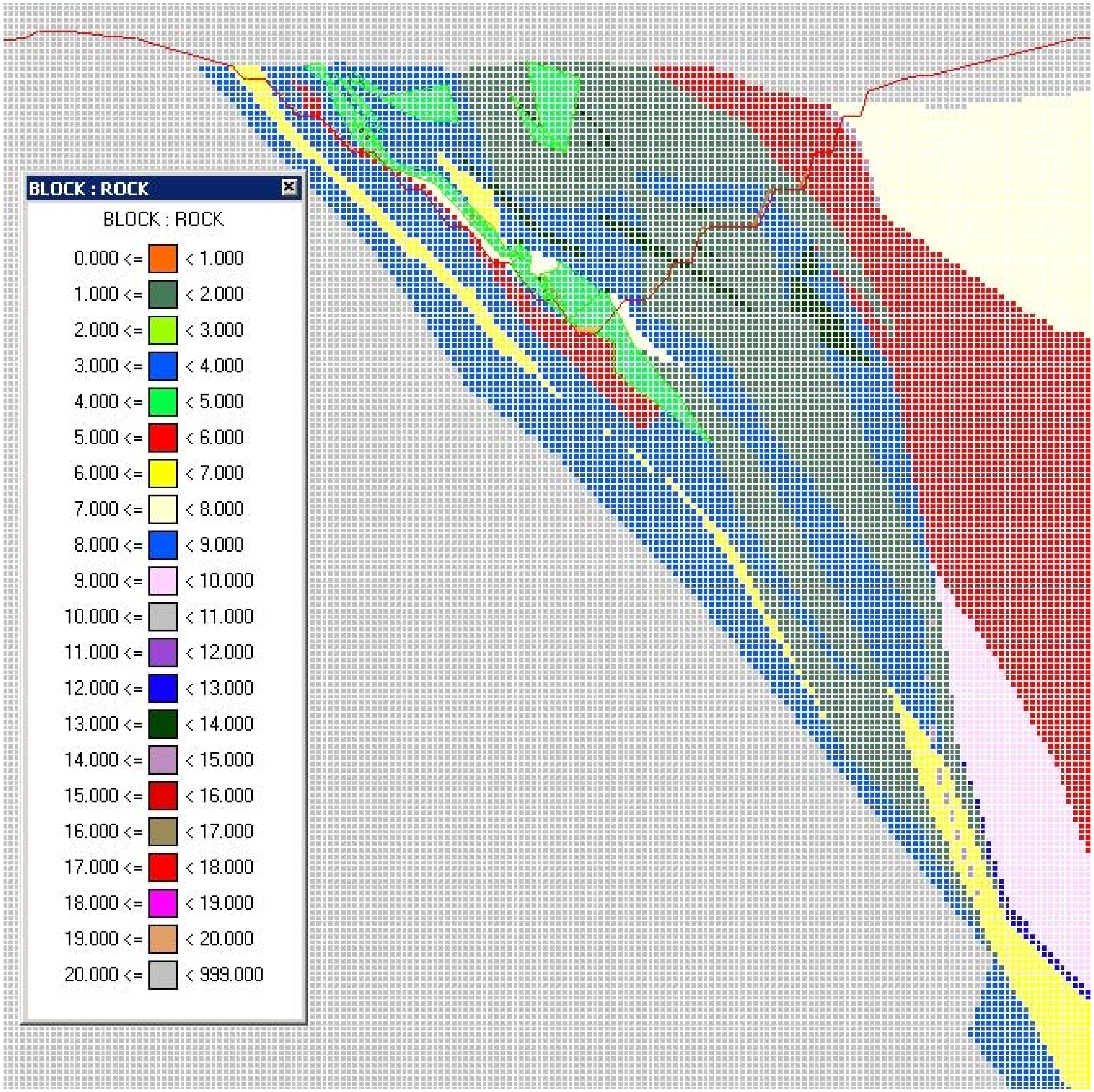

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ApolloGold™

BLACK FOX

TYPICAL VULCAN CROSS SECTION

SRK JOB NO.: 0144418.00 / TASK 0005

FILE NAME: 0144418.00.Fig.5-6.Typ.Vulcan.Cross.Sections.dwg

TIMMINS, ONTARIO, CANADA

DATE:	APPROVED:	FIGURE:	REVISION NO.
MAR. 2008	BS	5-6	A

6 Deposit Type (Item 10)

The Black Fox mineralization is an Archean age, lode gold deposit located within the Abitibi greenstone belt. The characteristics of this deposit type include; greenstone host rocks and gold-bearing quartz-carbonate veins. The veins occur as two main types. The first are arrays and stockworks along faults and shear-zones with a quartz-carbonate laminated fault-fill. The second are widely distributed extensional veins within carbonatized metamorphosed greenstone rocks. These deposits are typically associated with crustal scale compressional faults with a vertical extent of ≤2km and limited metallic zoning (Dubé and Geosselin, 2007).

The Black Fox deposit lies along the DPFZ, a major, east-west trending, deep-seated, crustal fault zone. The DPFZ and its numerous splays are associated with many past and current producing gold mines and gold deposits in the Porcupine Camp. The Stock and Aquarius gold deposits are located immediately west of Black Fox and the Holloway and Holt-McDermott Mines are located immediately to the east. Each of these deposits hosts approximately the 800k to 1Moz-Au. The Black Fox deposit is situated midway between two major mines, the Dome-Hoyle Pond and the Holt-Holloway. The Dome-Hoyle Pond deposits located within the same structural regime 65km west, have shown that gold bearing structures can be traced to 1,600m below surface where they remain open at depth. The Holt-Holloway Mine, located approximately 45km to the east has been developed down to 1,200m below surface.

There are several different styles of mineralization in the deposits associated with the DPFZ. The gold mineralization is structurally controlled, in a variety of geological settings. Alteration types include pyritic ankerite-sericite ± silica-albite altered mafic volcanics, green carbonate fuchsitic altered ultramafic volcanics with quartz stockworks, pyritic, porphyritic to syenitic felsic intrusives and multiple stages of quartz veins with free gold. Much of this variation is found at Black Fox (Prenn, 2006).

7 Mineralization (Item 11)

Gold mineralization at Black Fox occurs mainly within an ankerite alteration zone 1km along strike and 20m to 100m wide. This alteration envelope occurs primarily within komatiitic ultramafics and lesser mafic volcanics within the outer boundaries of the DPFZ. In some areas, the auriferous zones occur as concordant zones which follow lithological contacts and have been subsequently deformed to slightly discordant zones that are associated with syenitic sills. Other auriferous zones occur in quartz veins and stockworks discordant to lithology(Hoxha and James, 2007).

The three main styles of gold mineralization observed at Black Fox are:

- Low-sulfide mineralization associated with abundant quartz veining and quartz stockwork within strong ankerite-fuchsite altered ultramafic volcanic rocks;
- Mineralization hosted within mafic volcanic units associated with >5% pyrite and minor to moderate quartz veining; and
- Mineralization hosted by silicified felsic dikes.

The first style is low sulfide mineralization occurring within quartz-rich portions of the AUV and CGR rock types. This includes the green carbonate alteration of the "Main Zone". The typical host is the ankerite-fuchsite altered ultramafic volcanic rocks, commonly found throughout the DPFZ. Quartz veining and quartz stockwork show multiple phases of veining and structural episodes. This is illustrated by cross-cutting veins, chloritic slip surfaces in the quartz veins, and breccia textures. Visible gold is common in high-grade areas (Hoxha and James, 2007).

The second style of mineralization is hosted within mafic volcanic units coded as BMV or MV. This style is referred to as the "Flow Zones". It is typically associated with >5% fine-grained pyrite, minor to moderate quartz veining and a strong bleaching may be present. The quartz veins are typically parallel to foliation, and visible gold is characteristically absent. This style of mineralization is common in the footwall portion of the DPFZ. It has been tested mainly by the eastern part of the 235 Level underground drilling (Hoxha and James, 2007).

The third style of mineralization is hosted in silicified felsic bodies. These include both quartz-feldspar porphyries and finer grained units which are possibly syenitic in origin, Mineralization in the felsic units is associated with increased silicification, pyrite and some quartz veining all associated with a fracture foliation. In the middle and hanging wall portions of the DPFZ, felsic-hosted mineralization can be correlated from hole to hole over short distances. In the footwall portions, blocks and lenses of felsic material are encountered which do not correlate from hole to hole (Hoxha and James, 2007).

According to Hoxha and James (1998) there have been 15 separate mineralized structures identified within the ankerite envelope. The two main gold-bearing zones of their classification are the A1 at the hanging wall contact and the C0 located at the footwall contact. The other smaller zones located between these two generally have less continuity and width and represent parallel, mineralized shears and faults.

Previous underground mining indicates that sub-horizontal, mineralized bodies located within the "Main Zone", can be up to 15m thick and very high grade. This suggests that zones of dilation were produced during episodes of structural movements. The majority of the other mineralized zones and quartz veins are 1 to 5m in width (Hoxha and James, 2007).

At least three generations of structurally controlled quartz veining have been identified in the underground workings. Quartz veins and stockwork zones within the main mineralized envelope are concentrated along shear/fault zones. These structures parallel the main mineralized envelope suggesting they are responsible for the location and formation of the mineralization. The presence of sigmoidal vein structures, multiple quartz injections and re-sheared vein material with chloritic slips indicate complex and repeated structural movements during a cyclic brittle-ductile deformation period. In the quartz stockwork zones, gold mineralization can be erratic possibly related to certain vein sets carrying gold, whereas others are barren (Hoxha and James, 2007).

Prenn (2006) states that “Gold mineralization has been encountered in drill core at depths of 700m below surface to date and, since the host ankerite zone appears to continue further down, it is reasonable to expect that additional mineralization will be encountered with deeper drilling”

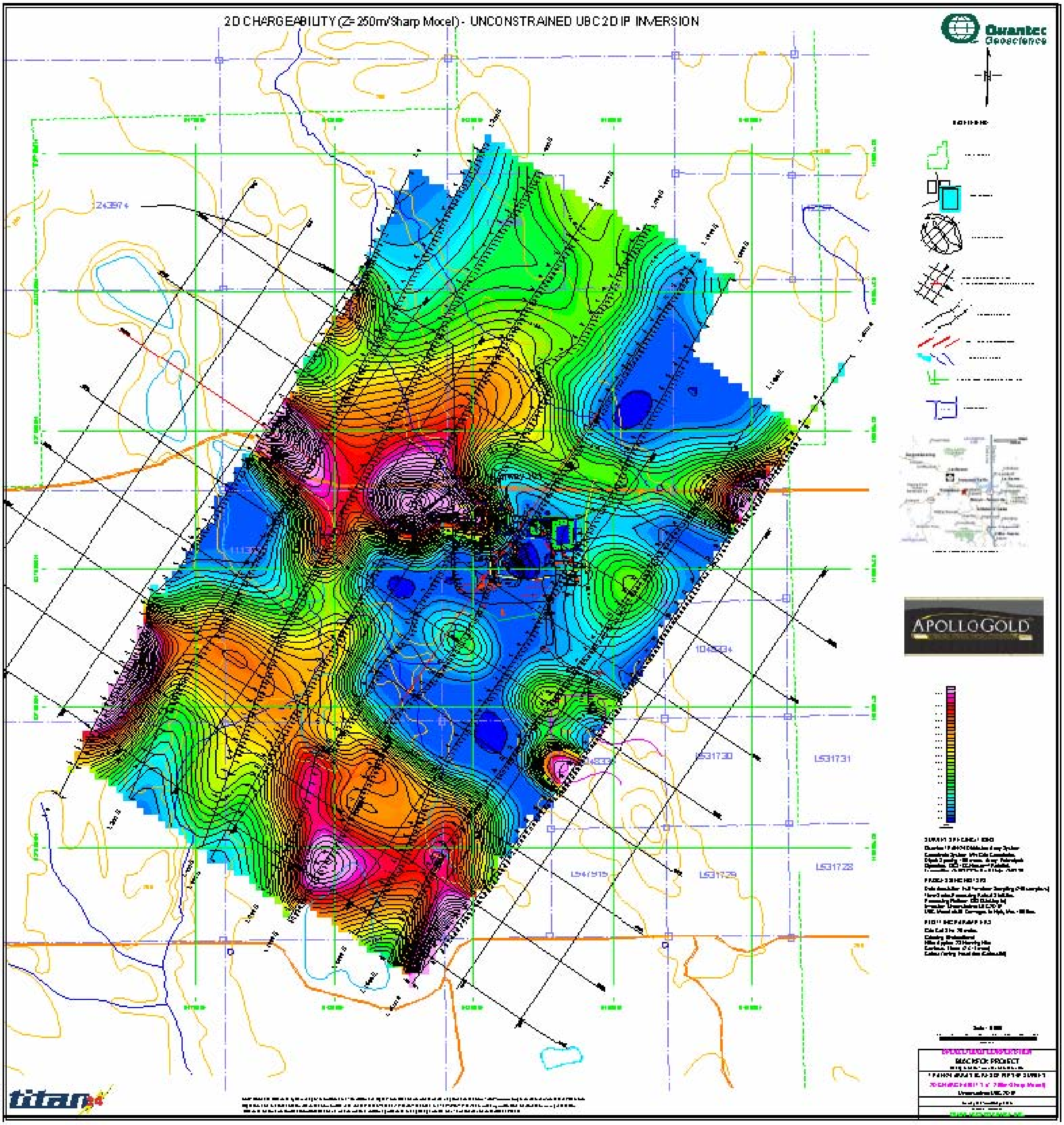
8 Exploration (Item 12)

This section is partly excerpted from the Technical Report Black Fox Project Matheson, Ontario Canada by N. Prenn of Mine Development Associates, August 14, 2006 and has been standardized to this report.

"The Apollo exploration drilling continued from previous campaigns on 12.5 to 25m fence lines. Two main emphases included, infill delineation of existing mineralization, and to explore for areas of new mineralization. In 2004, a 1,250m long exploratory underground drift (4m x 4m) was developed in the hanging wall down to 235m below the surface, to establish drill stations for an underground drilling program. The underground drilling program consisted of 75,700m of diamond drilling from 371 core holes. Surface drilling continued and by the end of 2006, Apollo had completed 825 diamond drillholes on the property, totaling 212,095m.

During the spring of 2003, Apollo Gold Exploration, Inc. contracted with Quantec Geophysical, Inc., Toronto, Ontario, to complete an IP survey covering the entire property. Lines were spaced every 200m with 100m dipole spacing. This survey has shown many chargeability and resistivity anomalies along both the DPFZ and the northwest projection of the Ross Fault. The Ross Fault is the host for the Ross Mine, located approximately 7,500m southeast of the Black Fox mine. In addition to these, a number of north-south trending anomalies were found. The intersections of these trends are considered to be prime exploration targets. It appears that the data from the earlier Noranda magnetic survey will also be valuable in defining exploration targets. The highly magnetic anomalies may help in mapping the basalt and ultramafic units on the property. In addition to this, low magnetic trends may be indicative of hydrothermal alteration that destroyed the magnetic qualities of the surrounding rocks. Figure 8-1 illustrates the results of the geophysical survey.

The initial portion of the Apollo surface drilling program concentrated on finding new ore zones below the Black Fox known Resources, along strike and adjacent to the known zones. The targets were the intersection of secondary faults with the DPFZ and also dilation zones within it. The mineralization is so tightly controlled by structures that a hole a few meters away could miss a high-grade zone. Fans of NQ-size drillholes were drilled to test for new ore shoots. The fans were spaced approximately 25m along strike and the intersections of the holes with the DPFZ were planned to be approximately 25m apart. The result of this program was the identification of a number of small, high-grade ore shoots that generally plunge at a 20^o to 40^o angle to the southeast or southwest, along the DPFZ. This is consistent with the intersection of two 45^o to 70^o dipping faults or with a zone of dilation along a fault that has both horizontal and vertical movement. Many of these ore shoots are still open with depth. A near-surface portion of high-grade mineralization was drilled on 12.5m spacing to improve the definition of this higher-grade mineralization".



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SRK Job No.: 144418

File Name: Figure 8-1.doc

Black Fox, Timmins, Ontario, Canada

Source: Mine Development Associates

Black Fox Geophysical Survey

Date: 07-10-07	Approved: DKY	Figure: 8-1

9 Drilling (Item 13)

9.1 Drilling Summary

A total of 1,889 surface and underground drillholes have been completed on the project by Noranda, Exall and Apollo between 1989 and 2008. Of these drillholes, 1,611 were completed by Apollo between 2002 and 2008. Table 9.1 lists the drilling by company and type. Figure 9-1 is a surface plan showing the surface drilling on the property. Figure 9-2 is a plan map showing the underground drilling. The Black Fox database includes 176,525 assay intervals.

Table 9.1: Black Fox Property Drill Summary

Company	Period	Type (All Core)	Number	Meters
Noranda	1989-1994	Surface	143	28,014
Exall	1995-1999	Surface	143	21,520
Apollo	2002-2007	Surface	500	146,684
Subtotal		*Surface*	*786*	196,218
Exall	1996-2001	Underground	707	61,115
Apollo	2004-2007	Underground	396	78,650
Subtotal		*Underground*	*1,103*	*139,765*
Total		**Black Fox**	**1,889**	**335,983**

9.2 Historic Diamond Drilling and Logging

Portions of this section were excerpted from Technical Report Black Fox Project Matheson, Ontario Canada by N. Prenn of Mine Development Associates, August 14, 2006 and have been modified and standardized to this report.

9.2.1 Noranda Drilling and Logging

Noranda drilled a total of 143 NQ-size diamond core holes between 1989 and 1994. The drillhole have an average depth of 197m and total to 28,014m. All holes were surveyed at the collar and had acid etch tests done to measure their dip angle. A Tropari survey was run at the bottom of a few of the deeper holes to measure deviation. The lack of down-hole surveys on many of the deeper holes will influence the accuracy of their location within the zone of mineralization. Core recovery was apparently very good as few recovery problems were listed in the logs. The core was brought to the surface and taken to Noranda's local logging facility. The core was logged for geology and geotechnical parameters.

9.2.2 Exall Drilling and Logging

Between 1994 through 1999 Exall drilled 143 NQ-size surface core holes totaling 21,520m and 707 underground core holes totaling 61,115m. All of the Exall drill core was NQ-size, unless ground conditions required reduction to BQ. The surface drillholes were down-hole surveyed, however, the underground holes were not surveyed for down-hole deflection, and therefore the bearing and inclination at the collar has to be used for the entire underground drillhole. The core was brought to a surface core area where the geologist logged and sampled it.

Exall resurveyed the collar coordinates of most of the Noranda drillholes, with generally good agreement in the coordinate conversion between the Noranda and Exall data.

9.2.3 Apollo Diamond Drilling and Logging

Norex Drilling International from Porcupine, Ontario, has completed most of the surface drilling at Black Fox for Apollo. The holes are typically NQ diameter core unless conditions require a reduction in core size. In general, ground conditions have been very good with average core recovery approximately 95%. The following sections document drilling, chain of custody and logging procedures employed by Apollo. Although no records are available to document the procedure used by the prior operators, there is no reason to suspect they did not follow standard industry practices of the time.

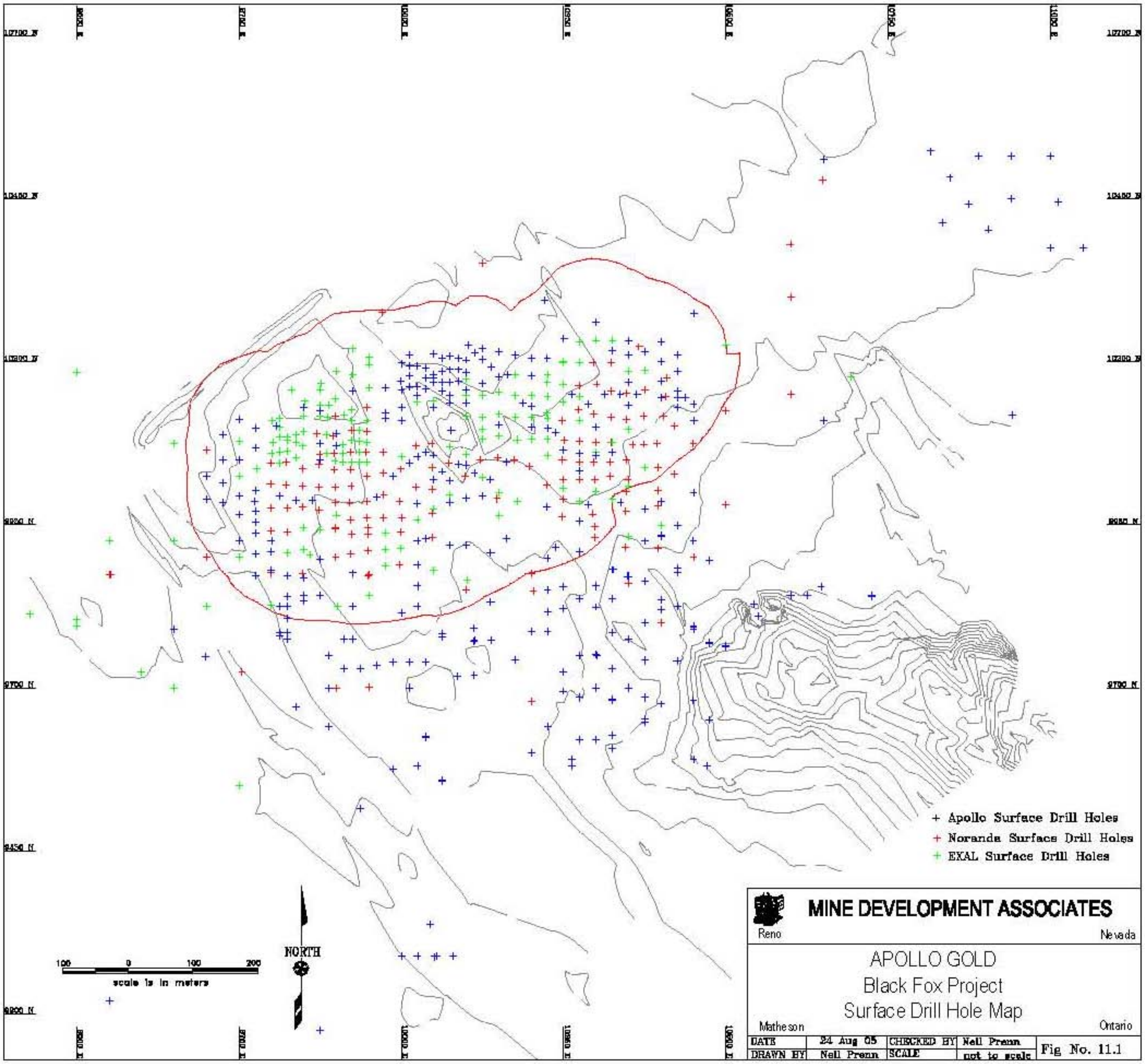
The core is removed from the wire line inner barrel and placed in wooden core boxes. Each box can hold up to 6m of NQ core. The depth at the end of the core run, along with the length of the run and the amount of core actually recovered, is written on wooden blocks, which are placed in the box at the end of the core run. When the box is full, the drillhole number, along with the beginning and ending depth is written on the outside of the box. A wooden lid is then placed on the box and the box is sealed with wire. The core is stacked at the side of the drill until it is picked up by representatives of Apollo Gold Exploration, Inc. During this time, the core is under the direct supervision of the driller.

The core samples are picked up by Apollo personnel each morning and at various times during the day as necessary. It is loaded into a company truck and taken to the core logging facility on the project site. The core is then unloaded from the truck, the wire ties are removed and the core is inspected for any damage that might have occurred during transport. Each box is then placed in racks within the core logging facility to await logging by Apollo geologists. When the geologist begins logging a hole, a logging form is first computer generated with data regarding the hole ID, depth, date logged, location and the logging geologist. All logging is done electronically with no handwritten data. This eliminates a separate data entry step and the subsequent errors that it can introduce. The geologist moves the boxes of core from the rack to the core logging table. The lids are removed and placed outside for later reuse. The pieces of core are then reassembled, within the box, just as it would have come out of the hole. The core is then measured and that measurement is compared to the core depth markers placed in the box by the drillers. This is documents core recovery and provides a check against any lost or missing core not accounted for by the drillers. All of this data, along with all geological data, are entered into the computer spreadsheet by the geologist. The core is then digitally photographed on the logging bench. This digital record is stored in the computer files for that hole. All of the geological information is backed up on the server daily.

Prior to removing the drill string, the downhole deflection is measured with a Reflex E-Z Shot digital tool (E-Z Shot). Measurements are taken approximately every 50m down the hole. Occasionally a spurious reading will be obtained near a particularly strongly magnetic rock unit. The geologists review all surveys and any such readings are discarded. As a check, three holes were re-surveyed using a Maxi-bore gyroscopic tool. The Maxi-bore survey duplicated the E-Z Shot survey very well. On average, the E-Z Shot gave readings that were within 3.1% on bearing and 0.4% on dip from the Maxi-bore survey information. All drillholes have their collars located by a licensed surveyor upon completion.

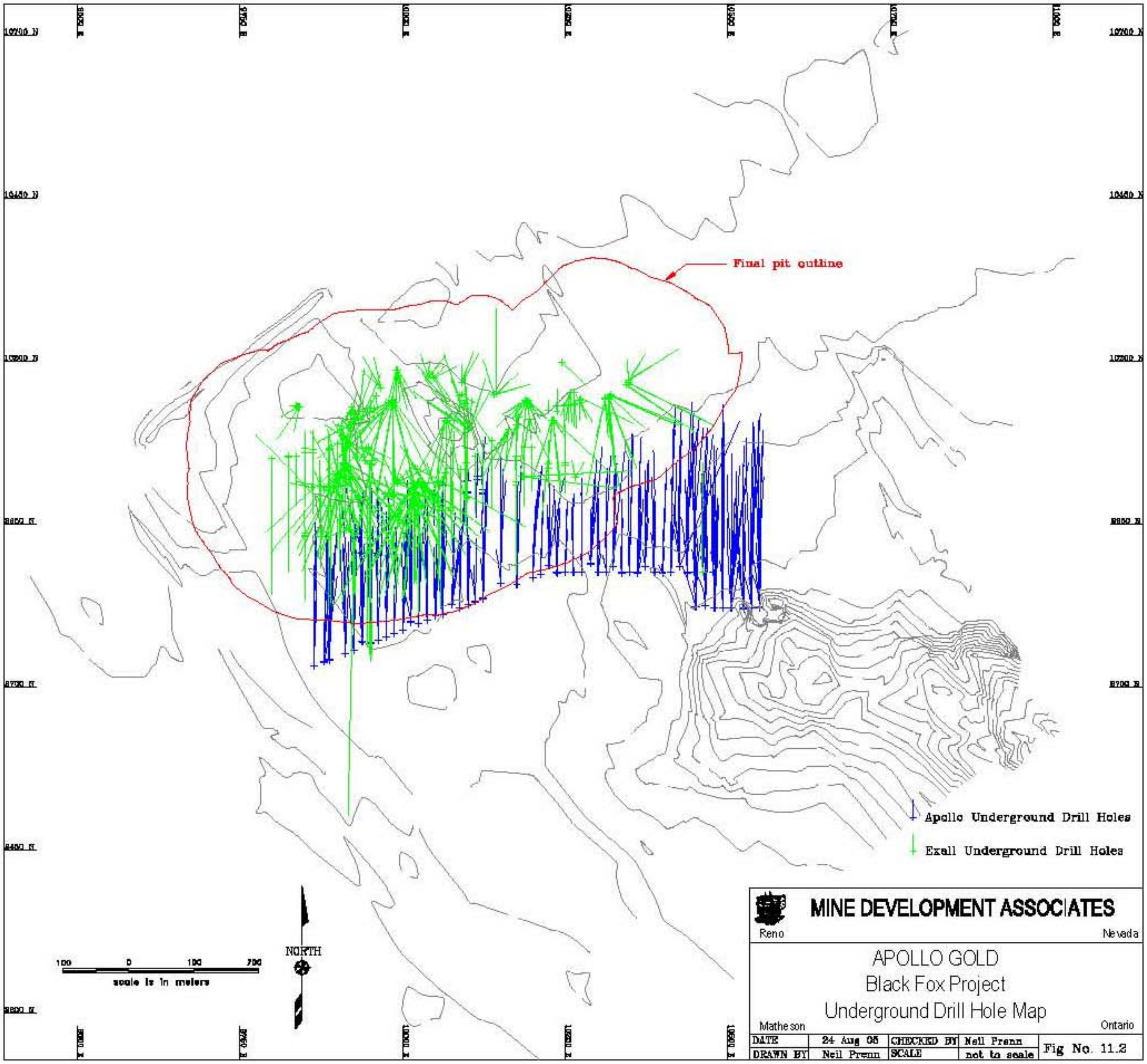
The Apollo drilling program has targeted two main areas of the mineralization. The first is the near-surface area where about half of the surface drillholes were completed. Drilling typically is located along sections oriented 036^{o} azimuth at inclinations of -45^{o} to -50^{o} to provide an alignment oriented nearly perpendicular to the DPFZ.

The second targeted area of mineralization is down dip of the previous drilling. At depth, the DPFZ has the same southeasterly strike, but the dip steepens to an average of -60^{o}. The mineralization still occurs along structural intersections and at dilation zones along the fault. These appear to rake at about -40^{o} to the southeast or southwest. In this area, the shoots tend to be smaller, thinner and less continuous than those encountered near the surface. The drillholes, which test this area, were collared from both the surface and underground. Typically, fans were used so that the structure was tested on 12.5 to 25m spacing. Eventually, more tightly-spaced drilling from underground platforms will be required to improve the delineation of the mineralization.



* Does not include 2007 Apollo Drilling.

SRK Consulting Engineers and Scientists	Black Fox, Timmins, Ontario, Canada	Surface Drill Plan		
SRK Job No.: 144418	Source: Mine Development Associates			
File Name: Figure 9-1.doc		Date: 07-10-07	Approved: DKY	Figure: 9-1



* Does not include 2007 Apollo Drilling.

SRK Consulting Engineers and Scientists	Black Fox, Timmins, Ontario, Canada	Underground Drillhole Plan Map		
SRK Job No.: 144418	Source: Mine Development Associates			
File Name: Figure 9-2.doc		Date: 07-10-07	Approved: DKY	Figure: 9-2

10 Sampling Method and Approach (Item 14)

This section is partly excerpted from Technical Report Black Fox Project Matheson, Ontario, Canada by Mine Development Associates, August 14, 2006 and has been standardized to this report.

10.1 Noranda Sampling

Little documentation is available describing the details of Noranda's sampling procedures. During the late 1980's it was not a standard component of project reporting to document the sampling procedures. Corporate standards of Noranda have always been to collect a representative sample. The core was logged for geology and geotechnical parameters and then cut in half with a diamond saw. The samples were then sent to either Swastika Labs or Chemex Labs in Rouyn, Quebec.

10.2 Exall Sampling

10.2.1 Exall Diamond Drilling

The core was brought to the surface where the geologist logged and sampled it. The core was split in half with a diamond saw. Prior to the installation of the mine site laboratory, Techni-Lab provided sample preparation of a 30g sample and completed a fire assay of the sample. All samples above 34.3gpt-Au were check assayed, as well as each 20th sample.

When the mine site laboratory was operational, they completed the analysis of the split core. Techni-Lab assayed the occasional overflow that the Exall lab could not handle.

10.3 Apollo Sampling

The sampling procedure begins with the geologist defining each sample interval and designating such with a sample tag documented in a sample book. They next mark the core with a center line cut mark and replace the core box lids for transfer to the sawing station. In the sawing room technicians saw the core sample in half with a diamond saw and place one half in a bag which is marked with the sample number and includes a sample tag. The half core that remains in the core box has the lid replaced and is placed back in the rack by the technician. Blank and standard samples are inserted approximately every twenty samples and are numbered in sequence with the core samples. The samples are then stored inside the core facility until they are picked up by Swastika Laboratories (Swastika) from Swastika, Ontario. The samples are placed into their truck, with each sample being checked off a list as it is being loaded and then taken directly to the laboratory where they are unloaded into a secure facility. At the logging area, once a truck load of split core has accumulated, the boxes are labeled with hole number and footage on stainless steel tags and then moved to, covered ,storage racks located outdoors.

10.4 Black Fox Deposit Sampling Issues

Prenn (2006) reports what MDA considers, two serious sampling issues at the Black Fox deposit. Both of which are related to coarse gold and sample size resulting in analyses that tend to report less gold than is actually present. The first issue relates to obtaining a large enough sample to represent the area it will influence. The gold at the Black Fox deposit appears to be concentrated in small areas causing drillhole samples to occasionally get too much gold in the sample or more commonly, missing the area of concentration and get too little gold in the sample.

The second issue relates to the particle size and distribution of the gold. When the particles are relatively large and not evenly distributed, the core holes can be too small to obtain a representative sample. This has a similar effect, in some cased it will over estimate the gold content but more typically underestimate it. Some samples may even appear to be waste having not encountered any gold particles that may be located relatively close by. It is likely that holes several meters in diameter would be required to obtain representative samples of the deposit. Prenn (2006) compared the areas that were mined with the drilling present and found many instances of drill indicated waste which were subsequently stoped.

This second issue is accentuated by getting the representative amount of gold in the sample pulp once the core sample has been split, crushed, split again and then pulverized. Gold particles up to 0.15cm have been observed and particles of 0.06cm are very common (Pitard, 2005). With gold this coarse, it is easy to create sub-samples that contain too many or too few gold particles if the sample size is not based on the size of the gold particles in the deposit. In order to sample the 0.15cm gold particles that occur at Black Fox, samples of up to 109kg must be processed in their entirety (Pitard, 2005). If the sample contains 0.06cm gold particles, which commonly occur in the deposit, a 7kg sample must be processed in its entirety (Pitard, 2005). These sample sizes are much larger than the typical 30g fire assay sample or even the generally larger than the 1,000g screen metallic assay sample. Once again, the samples result in a few assays containing too much gold, with far more containing less than is actually present in the whole sample.

Without proper size samples the database for the deposit likely contains a few samples that are too high in grade, but far more that are too low in grade. Francis Pitard concluded in his 2005 report on Black Fox mineralization that:

- *"The size of the core samples can account for local geology, but cannot account for the local gold content: Relative to the size of the coarse gold, the core mass is too small. The resulting effect is called the In Situ Nugget Effect: It is of the utmost importance for management to understand it;*
- *As a result, Poisson skewness enters the database, leading to a frequent under-estimation of many ore blocks, and an occasional over-estimation of a few ore blocks;*
- *Such skewness, if carried too far, as I believe is the case, can under-estimate the gold content of the deposit. However, and this is very important, it is an undeniable fact that the Ore Reserves are under-estimated. This is something to keep in mind: Poisson skewness affects the grade somewhat, but above all, makes a disaster on the estimation of the Ore Reserves, unless you are very lucky by having sharp, natural and obvious ore boundaries (e.g., Midas mine in Nevada); and*
- *By the time the sample is taken to the laboratory sample preparation, you have already lost its main purpose which is to be reasonably representative of all gold particle size fractions. Then, the preparation and assaying procedure, ignoring the potential presence of coarse gold, makes things even worse, most likely introducing a superimposed secondary Poisson skewness in the database."*

Prenn (2006) concurs with Pitard's (2005) conclusion, that the drillhole data is likely biased and will likely underestimate the contained gold within the deposit.

SRK concurs with the observations and opinions of Prenn (2006) and Pitard (2005) as discussed above. Based on these observations and opinions, SRK has put significant emphasis into creating the Resource estimate described below which approximates the historical production while remaining conservative.

11 Sample Preparation, Analyses and Security (Item 15)

11.1 Sample Preparation and Analyses

This section is partly excerpted from Technical Report Black Fox Project Matheson, Ontario Canada by Mine Development Associates, August 14, 2006 and has been standardized to this report.

11.1.1 Noranda Drill Sample Preparation and Analysis

The first phase of the Noranda drilling was processed by Min-En Laboratories Ltd. and TSL Laboratories (Holes 1-17). Noranda then used Swastika or Chemex Labs for analysis of the remainder of the samples. Noranda instructed the assay lab to prepare a 15g sample for analysis, and to re-run samples if the initial analysis was greater than 2gpt-Au using a 30g sample. The Noranda assay lab used the flowsheet shown in Figure 11-1 to prepare and assay the samples received from Noranda, most of which weighed from 1 to 5kg.

11.1.2 Exall Drill Sample Preparation and Analysis

Exall utilized Techni-Lab to complete the assaying of their drillholes until the mine site lab was completed. After completion of the mine site lab in February 1999, most of the assaying for the muck and chip samples was completed at the on-site laboratory, with Techni-Lab used for the drillhole samples, overflow and check assaying.

Techni-Lab dried and crushed the sample to 10 mesh, where a 300g split was taken. The 300g sample was pulverized to 80% -200 mesh. A 30g sample was split from the pulverized material for fire assay with AA finish. Exall requested checks on all assays exceeding 34.3gpt-Au. The Techni-Lab internal checks agreed well with the original sample.

11.1.3 Exall Mine Site Assay Lab Procedures

Blank samples were introduced with regular samples to verify the accuracy and to see if any contamination was present at the lab. Split assay pulps were sent to an external lab for comparison to verify the accuracy of the Exall mine site laboratory. From January 27 to February 25, 1999 a total of 370 samples were sent to Techni-Lab in Ste. Germaine Boule, Quebec. The difference between the Exall Lab and Techni-Lab was an average of 1.45%.

11.1.4 Apollo's Drill Sample Preparation and Analysis

Apollo saws the core and ships ½ of the drill core to either Swastika or SGS Laboratories. The labs prepare a 30g sample for fire assay with a gravimetric finish. The core is first crushed -10 mesh and a 400g split is then pulverized. As a quality check, the coarse reject sample material from each mineralized zone, over 1.0gpt is sent to the other lab. The rejects are re-split, pulverized and re-assayed using a 30g fire assay with a gravimetric finish. This procedure provides a check on the entire assay process, from sample prep through the gravimetric finish. Many of the higher grade samples are run with a screened metallic fire assay. All check data is subjected to a standard QA/QC analysis.

Swastika sends certificates of Analysis and electronic data files directly to the Apollo office in Matheson, Ontario. Hard copy results and assay certificate are also faxed to Apollo. The faxed certificates, are marked up with specific hole intervals and cross checked to the digital file for errors. After confirmed to be correct, the faxed copies are stamped complete, and added to the audit file for back referencing. The digital assay file is cut and pasted directly into the electronic core logs. Once the results are pasted in, the sample numbers are cross-referenced to ensure no pasting errors occurred. The completed drilling logs are then saved into a separate file. Once the logs are complete with all assays, they are saved as a “DC” file. The “DC” files are put into a locked folder on the Black Fox database, which can only be accessed as a read-only file. All editing of these files must be done through the Administrator (Project Manager). Once the file has been saved to this folder, the file is sent to Apollo’s offices in the USA for modeling and reporting purposes. If the assays results are not complete, the file will be saved as a “Pending” file, and is stored in an incomplete assays folder until final assay results are posted. All reported assays are final assays, and original certificates of analysis are stored in a separate binder and stored in a fire proof safe at the Black Fox mine site. All assay reporting goes through the Black Fox Project Manager.

11.2 QA/QC Analyses

11.2.1 Summary

During the development of the SRK pre-Feasibility Study, Analytical Solutions (ASL) of Toronto Canadawas contracted to provide an independent QA/QC review of historical and current sampling at Black Fox (Bloom 2006, 2007). The following paragraphs summarize their findings

ASL has been contracted to review documentation related to assay quality control and sampling for the Black Fox mine. The principal objective is to justify use of the existing assay database for Resource calculations. There has been considerable work by other consultants on the same subject and it is no intention to repeat the previous work.

The focus of the studies by ASL is to determine (a) whether there is any evidence of bias in the assay database and (b) the effect of coarse gold on the reliability of the assays.

The Black Fox assay data includes 128,026 assays. The 50th percentile for the dataset is 0.06gpt, the 90th percentile is 0.77gpt and the 95th percentile is 2.23gpt. It is apparent that only the upper 5% of the samples will influence the Resource calculation and the focus of the review should be this relatively small percentage of samples in the database.

No evidence has been found by previous consultants, who have done extensive reviews of procedures and data, of a bias in the gold assays. A systematic bias over a significant amount of time would affect a Resource calculation but this problem has not been identified.

Concerns have been raised regarding sample representivity of the Black Fox deposit. Thousands of pulp and reject duplicates confirm that it is difficult to reproduce assays within an arbitrary ± 10% but the assay reproducibility is typical of similar deposits and does not represent a material risk.

The historic check sampling on the project appears to be weak based on current QA/QC requirements for similar styles of gold mineralization. The Noranda check assays appear to be limited to only the same assay pulps. In general, they show reasonable agreement on the mean grade, however individual sample variance is relatively high. The Exall check assay program also was conducted on the same assay pulps. Techni-Lab, who conducted the majority of the exploration assaying for Exall, have been shown in a previous report to produce good reproducibility of the assay pulps.

Apollo has implemented a significantly improved check assay program where there is a check assay on each mineralized interval. In addition to the blank and standard check samples, Swastika runs its own internal check samples. All of the samples are run using a 30g fire assay. Relatively higher-grade zones are selected from the fire assay results by Apollo personnel and these intervals are re-run with a 1,200g screened metallic assay. Two of these samples are selected out of each ore zone at random and the rejects are sent to SGS Laboratories in Rouyn, Quebec where they are re-prepped and run for a second screen metallic assay. This is used as the quality check on the first assay set run by Swastika. All of the assay data is sent to Apollo in digital format where it is merged with the geological spreadsheet for that hole.

11.2.2 Noranda Check Assays

The Noranda data includes 196 reruns of 15g samples of the original 15g samples. The reruns average 4.6% lower grade than the original samples, as shown in Figure 11-2. The samples over 2g were noted to be rerun by a 30g sample, however most of this data is not in the digital database. Reruns of 80 samples indicate the reruns of 30g are higher in grade by about 5% than the original 15g sample, as shown in Figure 11-3. Evens (1997) of Roscoe Postle Associates Inc. (RPA) reports that Noranda checked about 10% of their assays.

The Noranda assay sample distribution is missing the high-grade found in all the other drill programs as shown in Figure 11-3. Prenn (2006) has recommended that check assays should be completed on the Noranda core that remains by metallic assay. These should be completed on intervals inside mineralized zones and just to the outside.

11.2.3 Exall Check Sampling

Techni-Lab batched samples in groups of 24. Each group contained at least one blank sample, one standard sample and duplicate samples. Routine checks were taken on about 5% of the samples and all samples over 34.3gpt-Au, however the check assay data is not present in the assay database. The statistics from past programs however are included in past RPA audits of the deposit Resources and Reserves for Exall. These indicated very good agreement between the Techni-Lab original assay and the Techni-Lab duplicate on thousands of checks of the same pulp.

11.2.4 Apollo Check Assaying

Metallic Check Assays

Apollo has completed screen metallic assays on 594 samples. Of these, 512 assays can be compared to normal fire assays. The screen metallic assays are 17% higher in grade than the average of the fire assays from these intervals. A total of 289 screen metallic assays are higher in grade to the average of the fire assays, while 223 are equal to or lower in grade. Prenn (2006) believes that screen metallic assays are essential in obtaining a sample assay that is more representative of the gold in the core sampled. Other assay methods will find too much gold on occasion, but the majority will find less than is in the core. Figure 11-4 shows the comparison of the metallic assays to the fire assays.

Standards and Blanks

Apollo submitted standards and blanks within each set of samples submitted for assay. Four labs were used with most of the assays completed by Swastika. Figure 11-5 shows that over several thousand tests that were completed the blanks typically agree.

A number of sample standards have been run within each group of samples. Figures 11-6 and 11-7 show the two most common high-grade and low-grade standards respectively. Swastika has reported reasonable ability to accurately assay the standards.

The following ranges were used to pass or fail the blanks and standards:

- Blank > 0.03gpt-Au = Fail;
- Standard 1.422 >1.528 or <1.322 = Fail;
- Standard 11.27 >12.03 or <10.63 = Fail; and
- Standard 9.62 >10.28 or <9.00 = Fail.

If the blank or standard failed, then the entire batch (20 samples) would be re-assayed, as well as the failed standard or blank.

Check Assays on Sample Pulps

A total of 8,425 sample pulps have been rerun by the original assayer. These samples indicate good agreement between the original sample and the rerun sample as shown in Figure 11-8. The check needed to be within ± 10%. If not, the pulp would be re-assayed a second time.

Checks on Sample Rejects

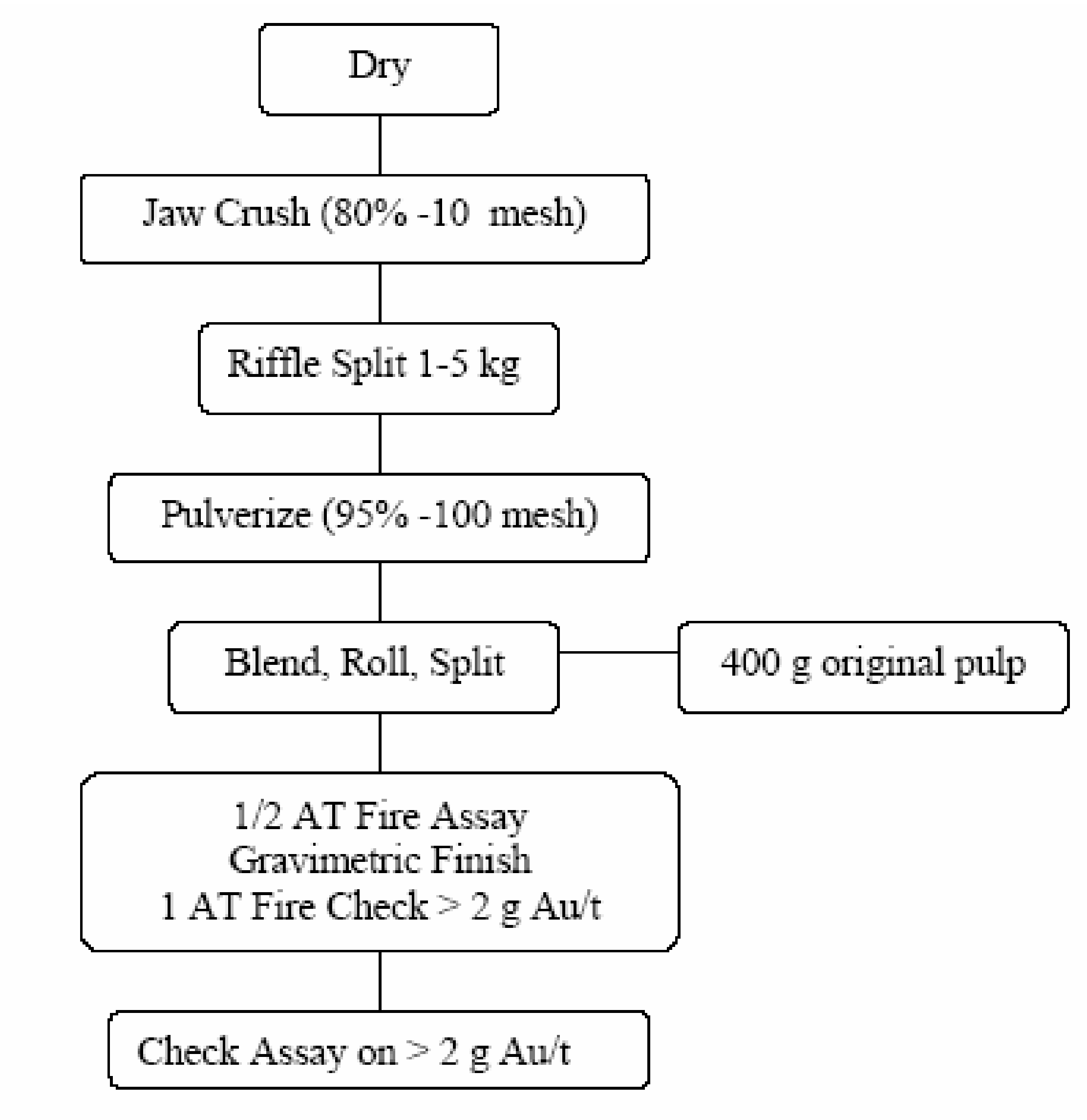
A total of 2,618 assay intervals have been checked by a different lab using splits from the sample rejects. The results indicate that the original sample is higher than the check by about 4%. This comparison is shown graphically in Figure 11-9. Of the 2,618 checks, a total of 905 or about 35% have differences of greater than 30%. If the checks were not within 20%, a second pulp would be prepared from the rejects. Figure 11-10 is a graph of the relative difference between the original and the checks. These differences are very significant and point out the need for a more substantial sampling and assaying program.

Mini-Bulk Sample Checks

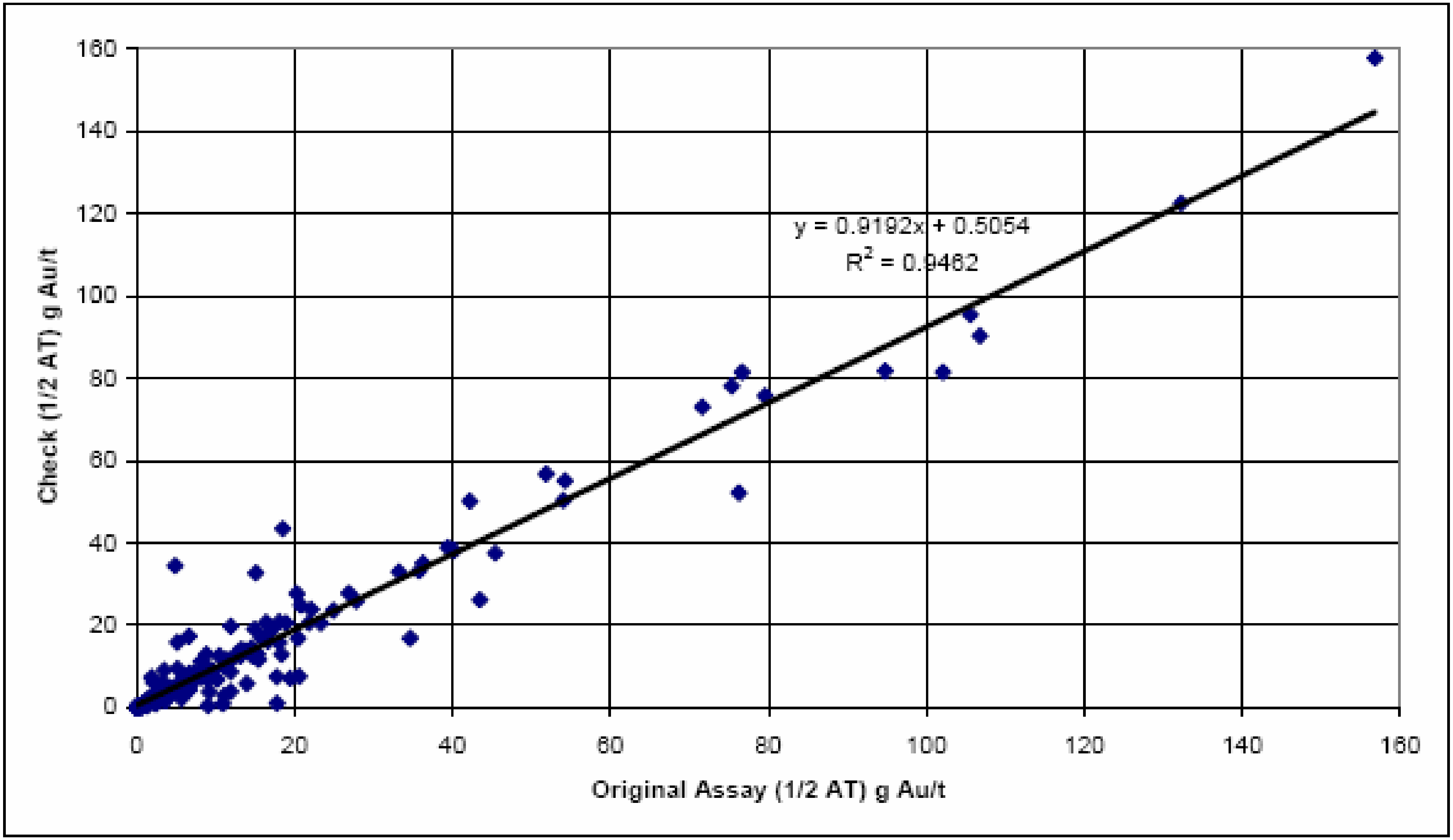
Large composites averaging about 14kg in weight were made by combining drillhole core and/or rejects. Typically, nine drillhole intervals were composited into one mini-bulk sample, however the range was 4 to 17kg. A total of 47 composites were made from mostly ore-grade intervals. Twenty-one of the 47 ore-grade composites contained high-grade. Since these tests use a much larger sample than the assay pulp, one would expect in a coarse gold deposit that the results of the mini-bulk sample gravity tests would be more reliable than the 30g pulps used for fire assay. The results of the 47 ore-grade mini-bulk gravity tests indicated a 9% lower grade in the mini-bulk samples compared to the individual assays. This is the opposite of what would be expected, and it is likely due to more high-grade material being in the mini-bulk samples than in the deposit as an average. Figure 11-11 shows the comparison of the original drillhole assays to the mini-bulk sample average grade. The six waste mini-bulk samples showed an improvement in grade of 1382.3% compared to the individual core assays. One of the waste samples averaged 0.00gpt-Au from the drillhole intervals and 2.82gpt-Au from the mini-bulk composites. The other five mini-bulk samples were not assayed prior to testing. One of these samples averaged 1.38gpt-Au from the mini-bulk test.

11.3 Summary

After the core was logged, the core samples were split by a diamond saw to obtain the assay lab sample. The 50% split was bagged at the site and either picked up by assay lab personnel or shipped to the assay lab. The sample was dried, crushed, split, pulverized, and blended to obtain fire assay pulps. The labs prepared 15g to 30g assay ton samples for assay. Most of the assays were completed by fire assay methods with a gravimetric finish.



SRK Consulting Engineers and Scientists	**Black Fox, Timmins, Ontario, Canada**	**Flowsheet for Swastica and Chemex Lab Sample Preparation and Assaying Procedure for Noranda**
SRK Job No.: 144418	**Source: Mine Development Associates**	Date: 07-10-07
File Name: Figure 11-1.doc		Approved: BAS
		Figure: 11-1



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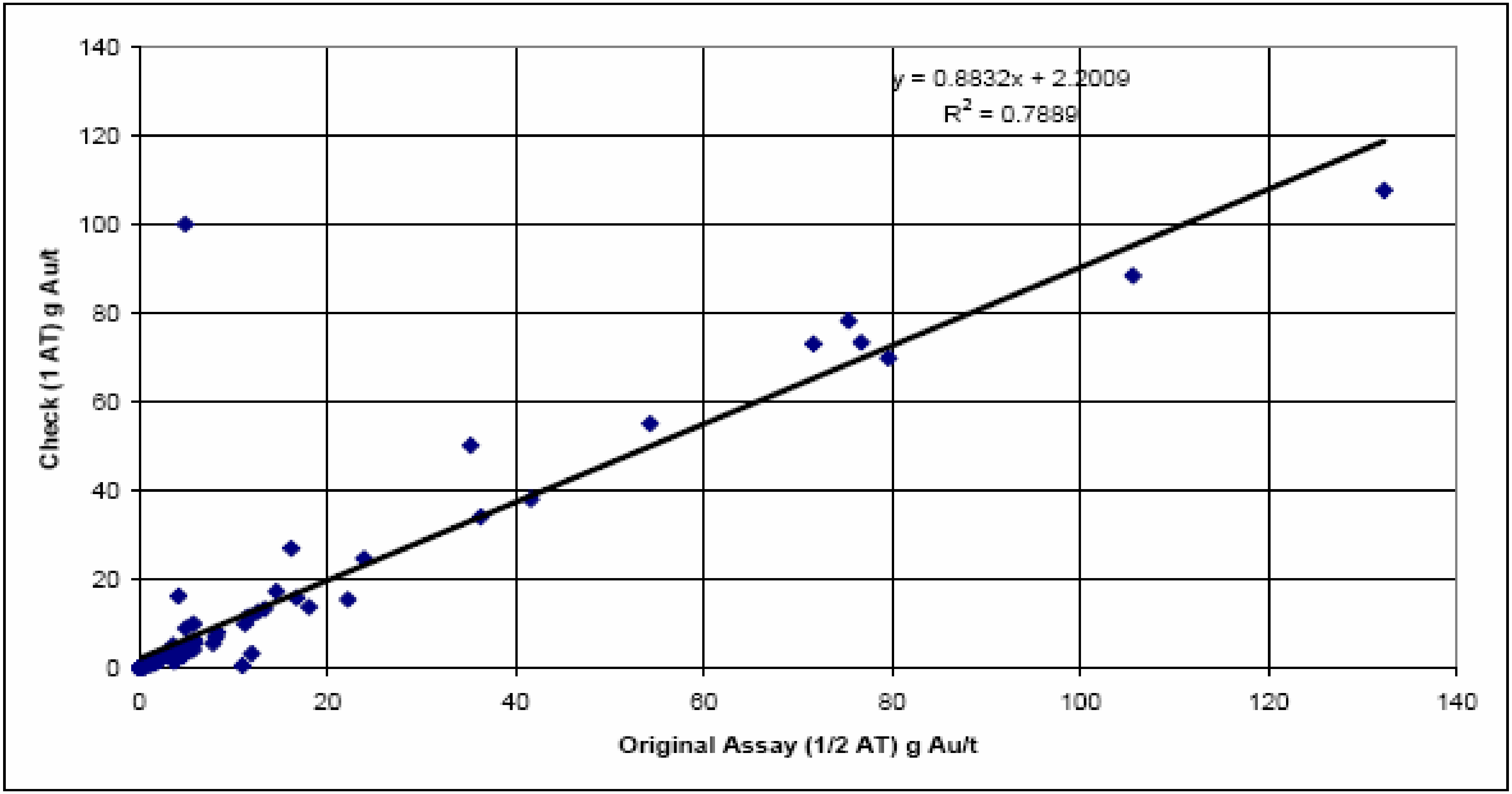
Noranda ½ Assay Tonne Reruns

Date: 07-10-07 | Approved: BAS | Figure: 11-2



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Black Fox,
Timmins, Ontario, Canada

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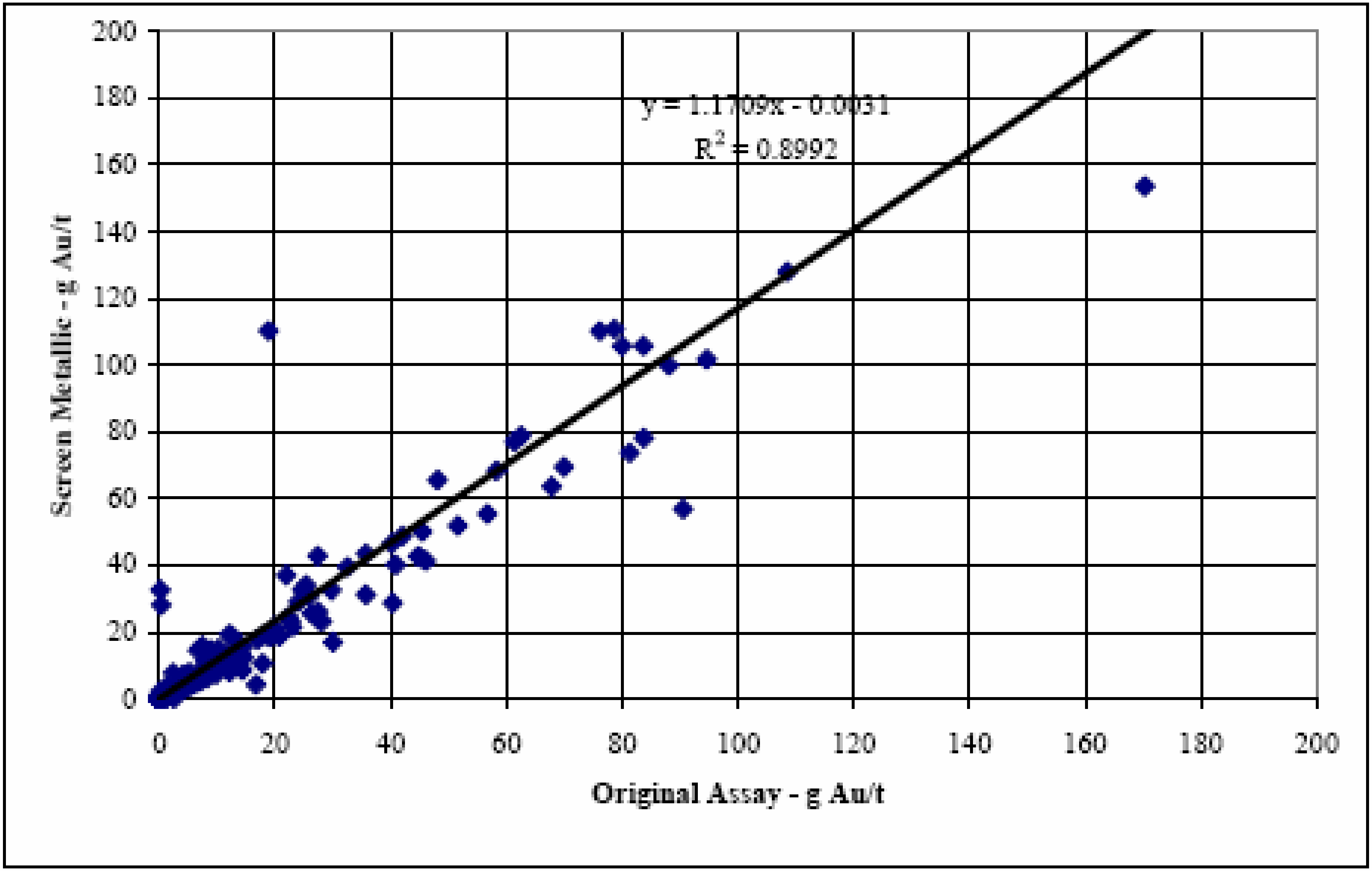
Noranda 1 Assay Tonne Reruns

Date: 07-10-07 | Approved: BAS | Figure: 11-3



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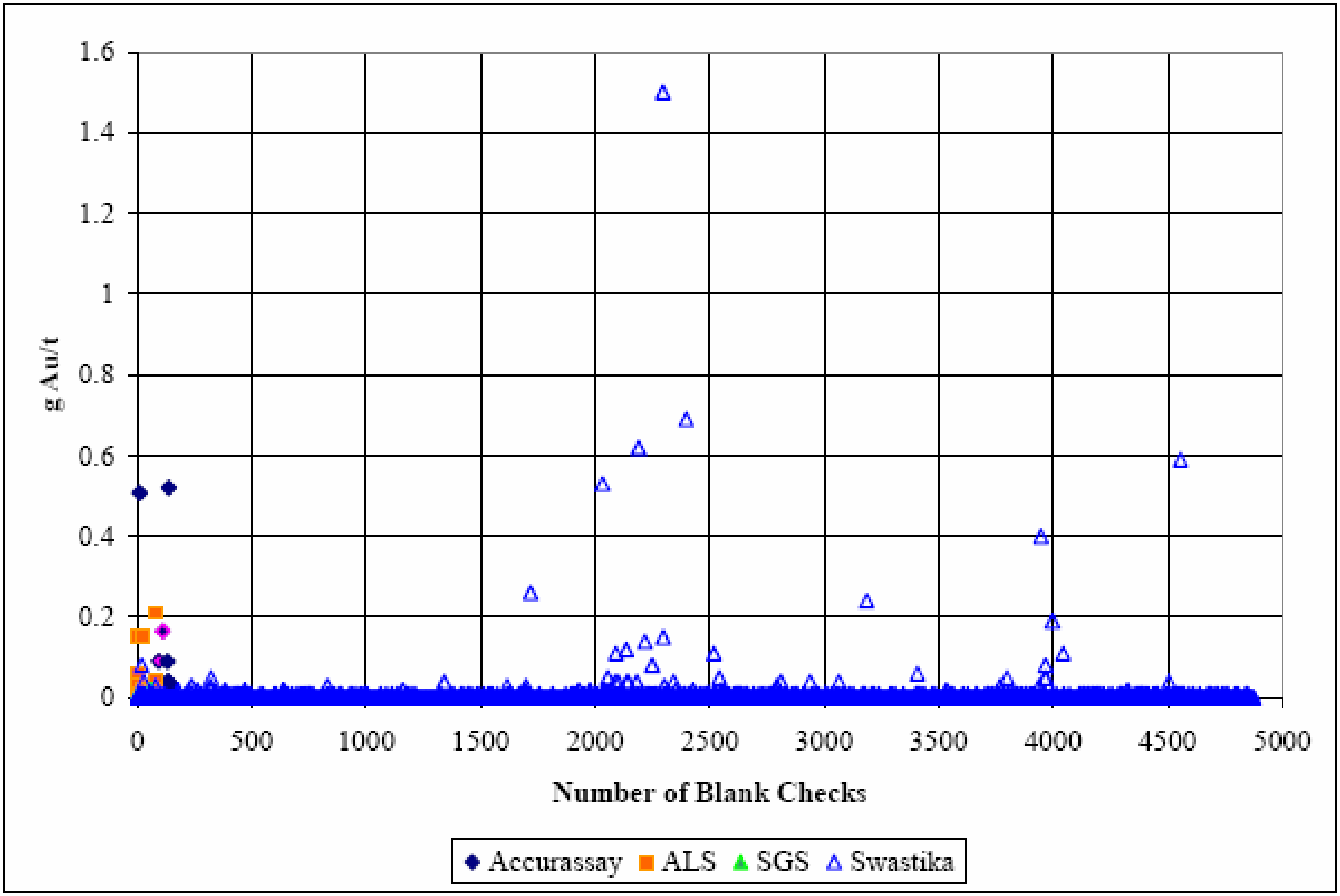
Apollo Metallic Check Assays

Date: 07-10-07 | Approved: BAS | Figure: 11-4



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File Name: Figure 11-5.doc

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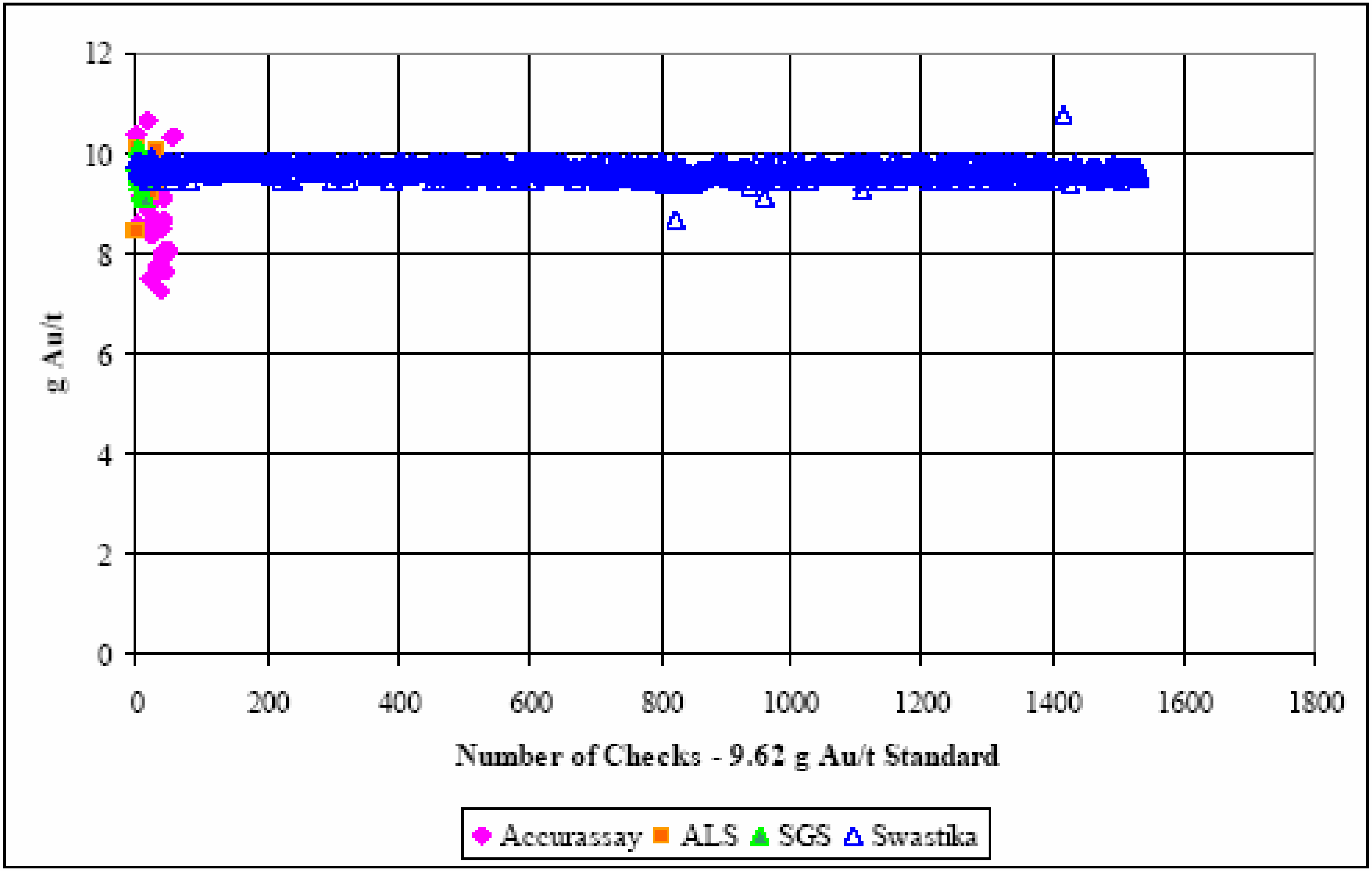
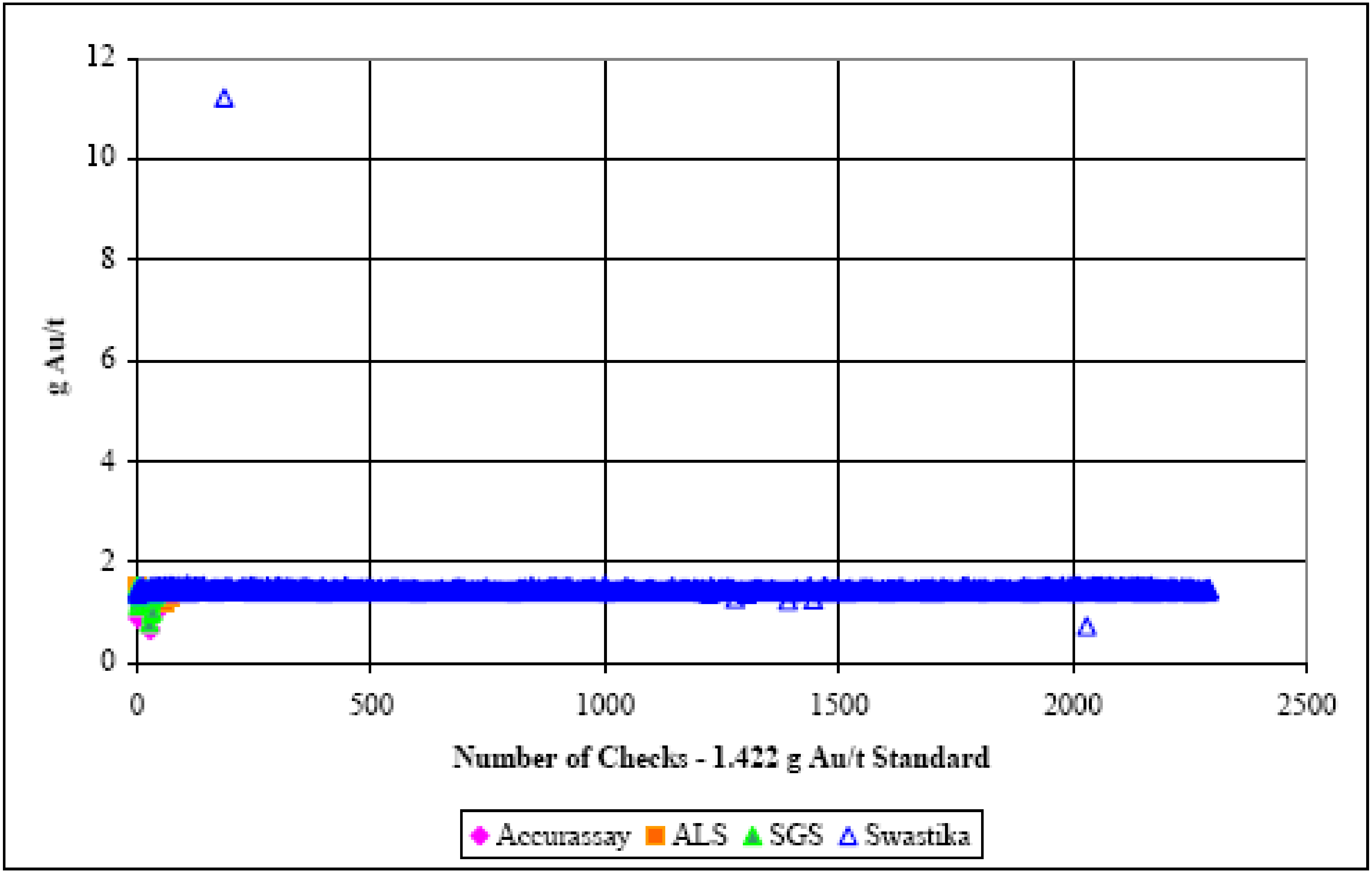
Source: Mine Development Associates

Apollo Blank Check Assays

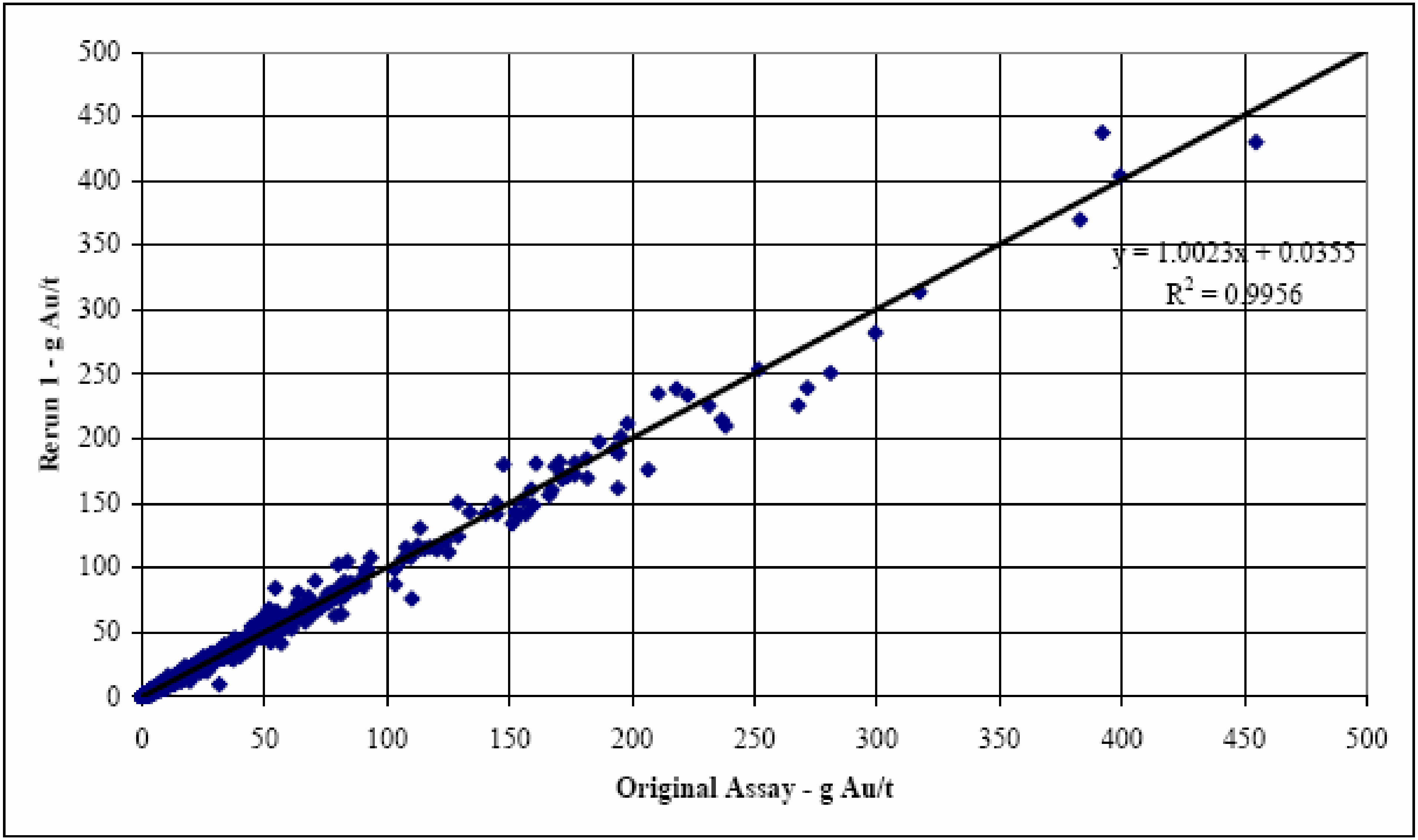
Date: 07-10-07 | Approved: BAS | Figure: 11-5



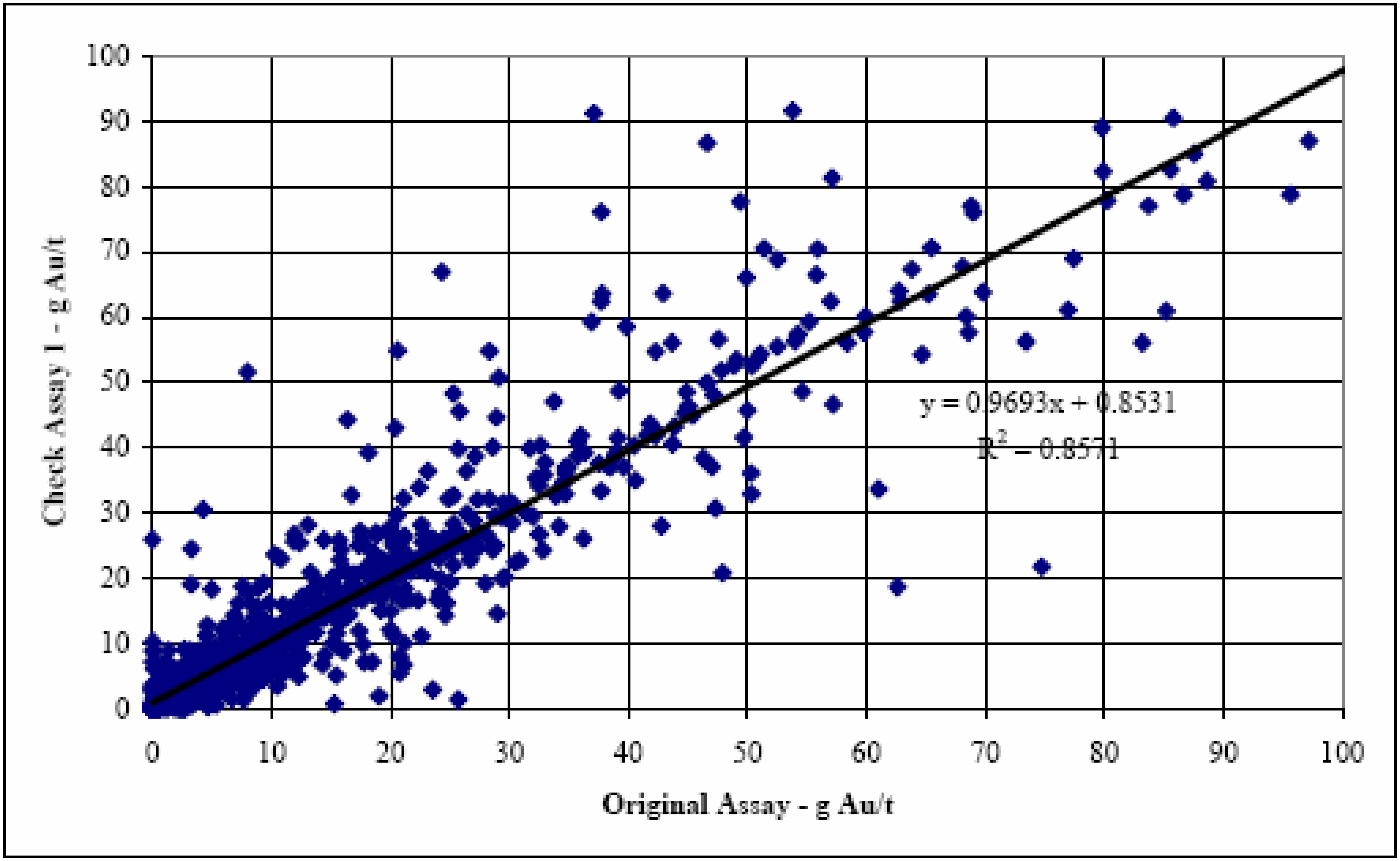
SRK Consulting Engineers and Scientists	Black Fox, Timmins, Ontario, Canada	**High-Grade Standard**
SRK Job No.: 144418	**Source: Mine Development Associates**	Date: 07-10-07
File Name: Figure 11-6.doc		Approved: BAS
		Figure: 11-6



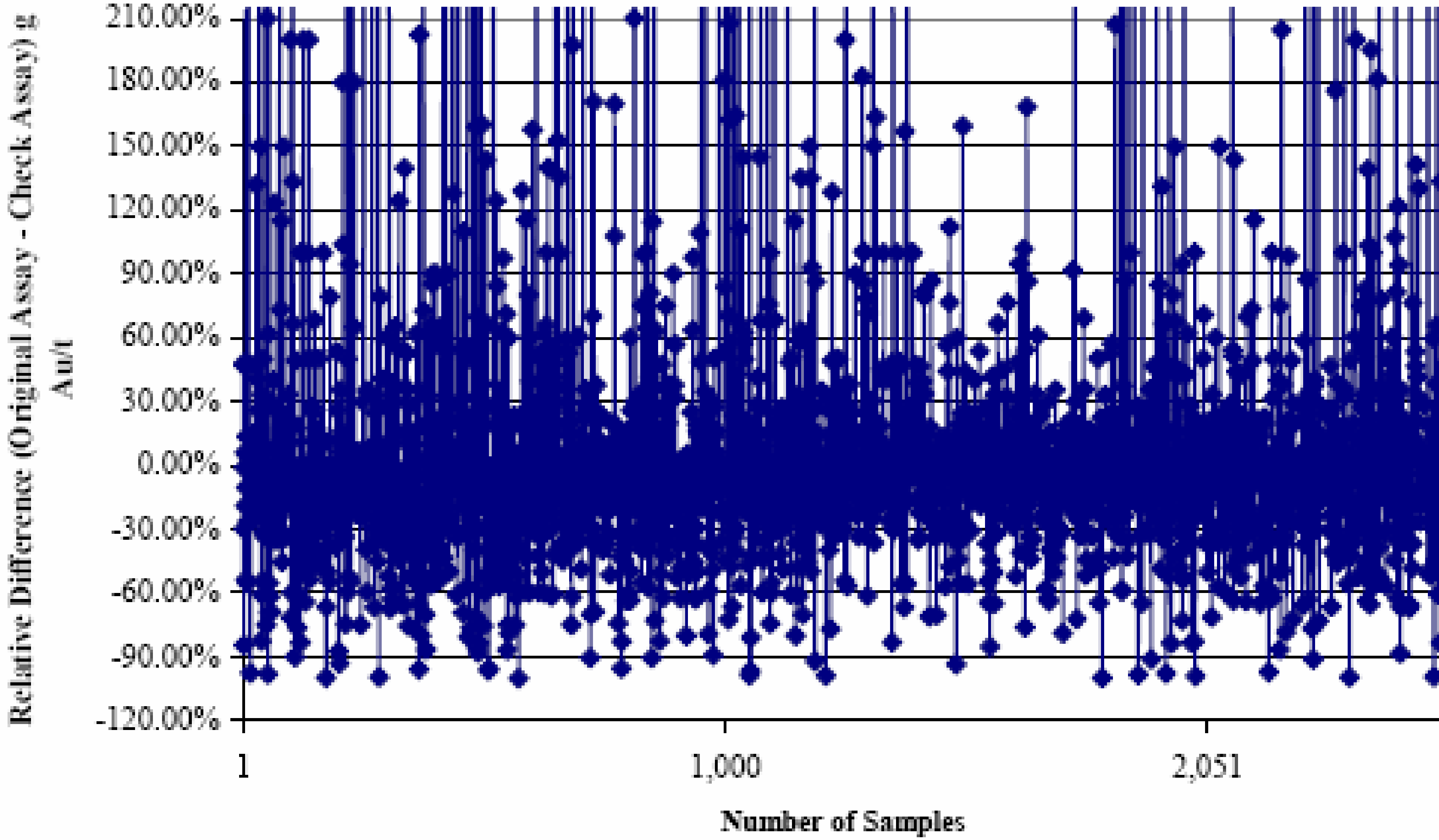
SRK Consulting Engineers and Scientists	Black Fox, Timmins, Ontario, Canada	Low-Grade Standard
SRK Job No.: 144418	Source: Mine Development Associates	
File Name: Figure 11-7.doc		Date: 07-10-07 Approved: BAS Figure: 11-7



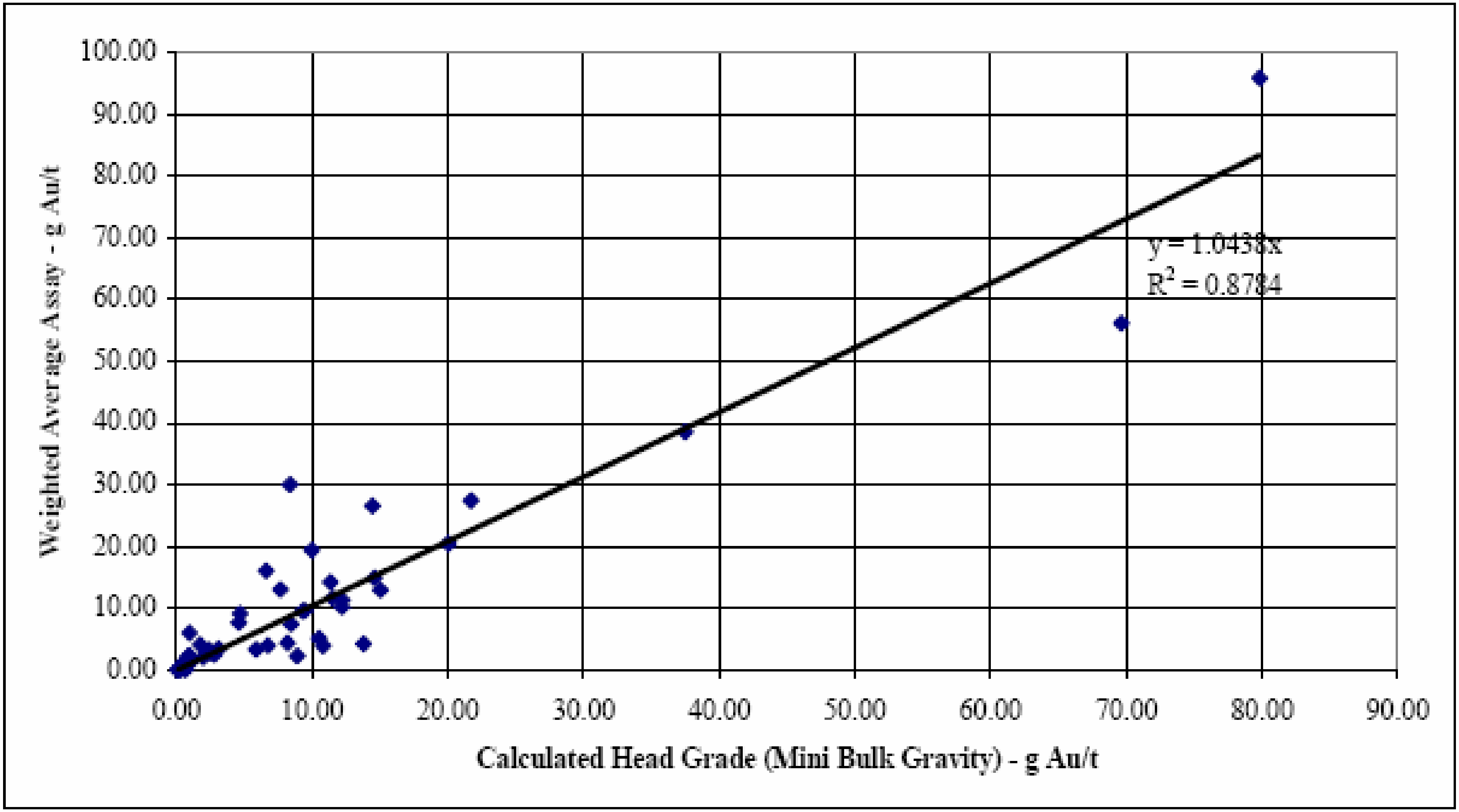
SRK Consulting Engineers and Scientists	**Black Fox, Timmins, Ontario, Canada** **Source: Mine Development Associates**	**Sample Pulp Check-First Run**
SRK Job No.: 144418		
File Name: Figure 11-8.doc		Date: 07-10-07 Approved: BAS Figure: 11-8



SRK Consulting Engineers and Scientists	Black Fox, Timmins, Ontario, Canada	Original Assay vs. Check Assay from Sample Rejects		
SRK Job No.: 144418	Source: Mine Development Associates			
File Name: Figure 11-9.doc		Date: 07-10-07	Approved: BAS	Figure: 11-9



SRK Consulting Engineers and Scientists	Black Fox, Timmins, Ontario, Canada Source: Mine Development Associates	Relative Difference-Original Assay vs. Check Assay form Sample Rejects
SRK Job No.: 144418		Date: 07-10-07 / Approved: BAS / Figure: 11-10
File Name: Figure 11-10.doc		



SRK Consulting Engineers and Scientists	Black Fox, Timmins, Ontario, Canada	**Mini-Bulk Sample Gravity Test Result**
SRK Job No.: 144418	**Source: Mine Development Associates**	Date: 07-10-07 / Approved: BAS / Figure: 11-11
File Name: Figure 11-11.doc		

12 Data Verification (Item 16)

Data verification at Black Fox consists of two primary areas of focus. The first pertains to verification of assay results obtained from an individual sample at a particular lab. The second pertains to verification of the numerical values contained in the electronic database to those reported on the original hard copy assay certificate.

The historic check assaying conducted by previous operators Exall and Noranda are considered substandard by today's requirements. However, the QA/QC study discussed above indicates that this data set presents no material risk to the current Resource estimation. Prenn (2006) reviewed Apollo's individual assay verification program and provided the following synopsis.

Apollo's program for data verification is a considerable improvement of the past checks, however, while the number of checks have improved, the sampling problems have become more evident. The metallic assays have shown a grade improvement of about 17% over the average of the fire assays for the same intervals. Check assays from pulps have shown good agreement with the original assays, while new pulps prepared from rejects have not shown good agreement with the original assays. MDA believes that the samples from drilling contain less gold than is representative from the area drilled, and that the fire assay samples contain less gold than is in the core sample.

MDA recommends that Apollo consider using metallic assays as the only appropriate method to sample the core, and that additional mini-bulk gravity tests and full scale bulk samples be completed for the main types of mineralization in the deposit.

SRK agrees with the recommendations of Prenn (2006) presented above but recognizes that screen metallic assays are quite expensive and typically provide a slow turn around time. One problem associated with a change in assay procedures at this stage of the project is that is would require a re-assay of as many pulps as are available in order to standardize the database. The benefits of such a program may not outweigh the time and cost associated with it. Considering that Apollo currently intends to move forward with a test mining program contingent on a positive Feasibility Study, model reconciliation with actual mining will provide a valid method to verify the proper usage of the assay data in the estimation technique. Additionally, ASL has reviewed the data set subsequent to the Prenn (2006) review and has concluded that it presents no material risk to the Resource estimation.

The second aspect of the data verification pertains to comparison of the numerical values contained in the electronic database to those reported on the original hard copy assay certificates. This work has been conducted by several reviewers and no problems have been reported to date. The original database was validated by RPA and subsequently by Prenn (2006). Further to this work, ASL cross-checked portions of electronic database to original certificates and reported no issues. SRK has provided independent verification of the electronic database subsequent to previous reviews in two ways. The first was to evaluate the current procedures used by Apollo to transfer the assay results obtained from the lab to its electronic database. SRK notes that these procedures provide adequate safeguards to the integrity of the assay database and meet or exceed current industry standards. Additionally, SRK was provided with original signed assay certificates from recent drilling and conducted spot checks comparing the certificate values to the electronic database and no errors were found.

13 Adjacent Properties (Item 17)

The Black Fox Project is located in the eastern side of the Porcupine District approximately 75km east of the Timmins Gold Camp. The Project is situated along the DPFZ, which hosts many important properties in the district (Prenn, 2006). This includes the Dome Mine, now part of the Porcupine Joint Venture, located in South Porcupine near Timmins, Ontario and approximately 65km west of the Project area. Properties proximal to the Project area include the Clavos, Hislop, Holloway, Holt and Taylor held by St. Andrew Goldfields and Ross held by Preston Electrical and Mechanical.

The Ross deposit is an underground mine that last operated in 1989. The reserve estimates, summarized in Table 13.1, was reported in 1989 and has been publicly disclosed. This historical reserve estimate was not estimated in compliance with NI 43-101 procedures and should not be relied on.

Table 13.1 lists reserves and Table 13.2 lists resources for selected properties. Adjacent properties are shown on Figure 13-1. The information on adjacent properties has been compiled from the Metals Economic Group (MEG) website. The qualified person has been unable to verify this information on adjacent properties and the information on adjacent properties is not indicative of the mineralization at the Black Fox Project.

Table 13.1: Reported Reserves from Selected Properties along the DPFZ

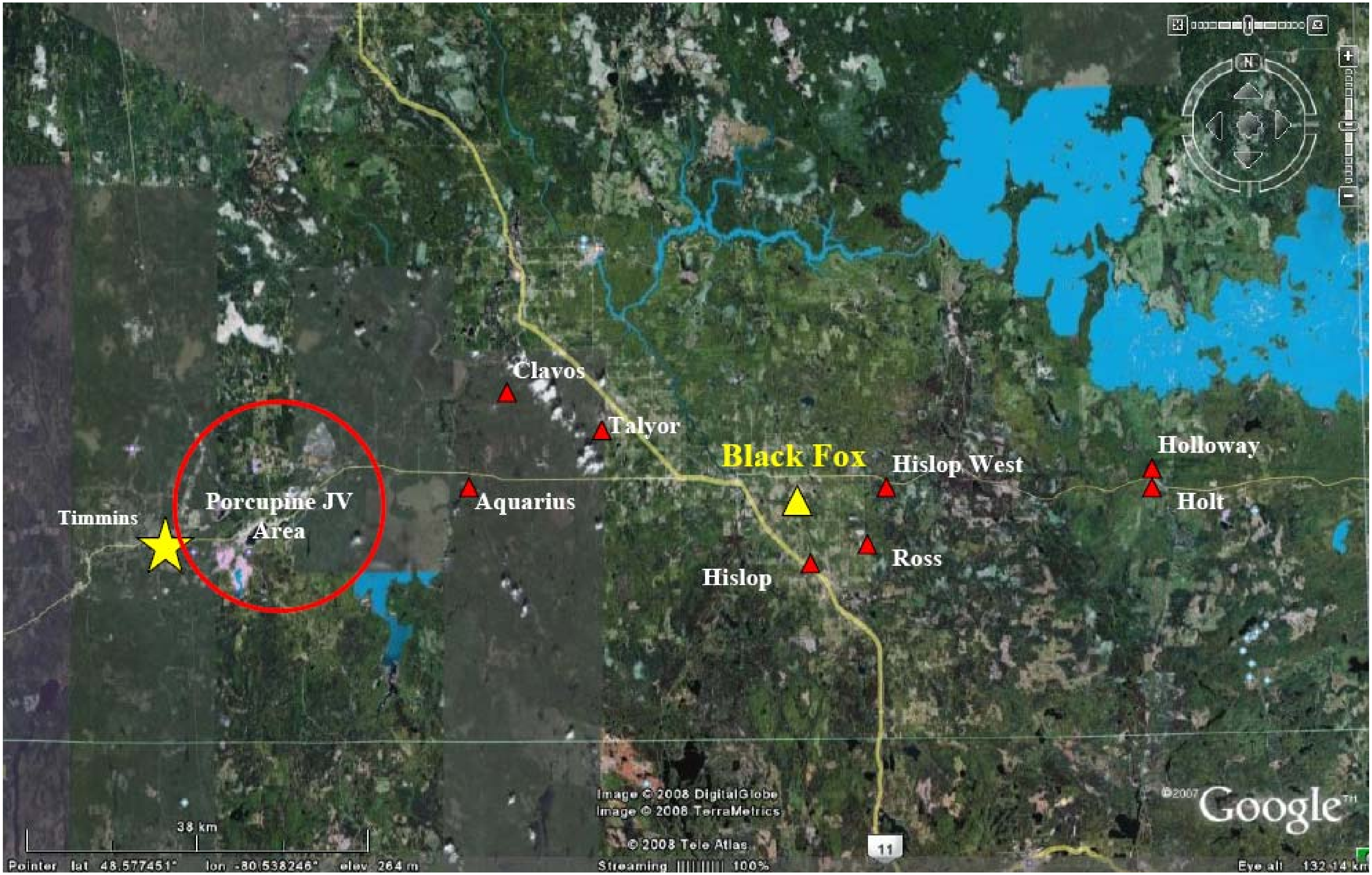
Company	Deposit	Proven			Probable			Proven and Probable		
		kt	Grade gpt-Au	koz-Au	kt	Grade gpt-Au	koz-Au	kt	Grade gpt-Au	koz-Au
St Andrew Goldfields	Aquarius									
St Andrew Goldfields	Clavos (Stock Complex)									
St Andrew Goldfields	Hislop									
St Andrew Goldfields	Holloway									
St Andrew Goldfields	Holt									
Kirkland Lake Gold Inc.	Macassa–South Mine Complex	46.8	25.0	38	390.5	25.4	319	437.3	25.39	357
GoldCorp Inc.	Porcupine Joint Venture: *Hoyle Pond, Pamour, Holinger Pit, Nighthawk Lake, Coniaurum, Hoyle North Shumacher, Delnite, and Timmins Division including Paymaster Shaft and Dome*							58,670	1.53	2,886
Preston Electrical and Mechanical	Ross*							630	4.35	88
St Andrew Goldfields	Taylor									
							Totals of Selected Mines	**59,737.3**	**1.73**	**3,331**

Source: Metals Economic Group, 2008. * This historical reserve estimate was not estimated in compliance with NI 43-101 procedures and should not be relied on.

Table 13.2: Reported Resources from Selected Properties along the DPFZ

Company	Deposit	Measured			Indicated			Measured and Indicated		
		kt	Grade gpt-Au	koz-Au	kt	Grade gpt-Au	koz-Au	kt	Grade gpt-Au	koz-Au
St Andrew Goldfields	Aquarius				23,112	1.49	1,121	23,112	1.49	1,121
St Andrew Goldfields	Clavos (Stock Complex)	26	7.83	6.5	117	8.12	31	143	8.16	37.5
St Andrew Goldfields	Hislop	192	5.82	36	107	14.05	48	299	8.74	84
St Andrew Goldfields	Holloway							1,037	7.8	260
St Andrew Goldfields	Holt							2,985	7.411	873
Kirkland Lake Gold Inc.	Macassa–South Mine Complex	18	8.2	0.475	190	23.3	142	208	21.305	142.475
GoldCorp Inc.	Porcupine Joint Venture: *Hoyle Pond, Pamour, Holinger Pit, Nighthawk Lake, Coniaurum, Hoyle North Shumacher, Delnite, and Timmins Division including Paymaster Shaft and Dome*							73,380	1.66	3,916
Preston Electrical and Mechanical	Ross									
St Andrew Goldfields	Taylor				1,405	7.6	343	1,405	7.6	343
							Totals of selected properties	**102,569**	**2.06**	**6,777**

Source: Metals Economic Group, 2008.



SRK Consulting
Engineers and Scientists

SRK Job No.: 144418

File Name: Figure 13-1.doc

Black Fox,
Timmins, Ontario, Canada

Sources: MEG and Google Earth 2008, modified by SRK

Aerial Photograph Showing Adjacent Properties

Date: 03/19/08 | Approved: DKB | Figure: 13-1

14 Mineral Processing and Metallurgical Testing (Item 18)

The feasibility Study considered three milling options for the treatment of open pit and underground ores from Black Fox:

- Holt Mill – Toll milling up to 2,500tpd ore at St. Andrews Goldfields' (SAS) Holt Mill,
- Stock Mill – Milling up to 1,100tpd at the Stock Mill currently owned by SAS but the subject of a letter of intent signed in March 2008 between Apollo and SAS for Apollo to purchase the mill from SAS.
- Black Fox Mill – Design build and operate a 1,500tpd mill at the Black Fox mine.

Given these alternatives, it was determined that the best option for Apollo Gold, given the current understanding of the Black Fox deposit is to operate the Stock Mill at full capacity (nominal 1,100tpd) and toll mill the remaining mine production (nominal 1,400tpd) at the Holt Mill.

The Black Fox Mill, while designed to feasibility-level, will be kept on reserve, until a time when additional reserves are discovered at Black Fox or when other, currently unknown, conditions make the construction of the Black Fox Mill economic.

14.1 Metallurgical Testing

This section excerpted from Technical Report Black Fox Project Matheson, Ontario, Canada by Mine Development Associates, August 14, 2006 and has been standardized to this report.

"The material in this section was developed mainly by Peter Taggart and Associates in conjunction with developing a metallurgical flowsheet and estimated capital and operating costs for a 1,500tpd processing plant used in the 2004 pre-Feasibility Study completed by MDA for the Black Fox Project.

Canadian Mine Development, commissioned by Exall to prepare a Feasibility Study, retained Mr. Rick Swider, Richard C. Swider Consulting Engineers Limited to direct metallurgical testwork performed by Lakefield Research Limited (Lakefield). The test program, conducted in 1996, was designed to assess the amenability of the Stock mill to treating the Hislop-Beatty mineralization. The comprehensive program confirmed the suitability of the plant and custom milling operations commenced in 1997.

In 1999, Kinross Gold was holding the Macassa plant in Kirkland Lake, on a "care and maintenance" basis. Exall elected to use this plant, upon the expiry of the three-year custom milling agreement with St. Andrew Goldfields. Operations commenced at the Macassa plant in October 1999 and were terminated in May 2001.

Exall commissioned Richard Swider to oversee additional bench scale and pilot plant test programs in 1999 to examine alternative process options that could enhance process efficiencies.

14.1.1 Mineralization at Black Fox

The Black Fox mineralization is hosted in two zones, the West Zone and the East Zone. The West Zone material principally comprises green carbonate and contains gold in quartz ankerite-veinlets. Minimal amounts of sulphide are present. The East Zone contains up to 5% sulphides, principally as pyrite. While the East Zone mineralization is slightly more refractory than the West Zone material, both exhibit free milling characteristics and yield gold recoveries in excess of 95%.

The mineralization contains finely disseminated visible gold and is amenable to gravity concentration. The host rock contains no graphite or cyanide consuming minerals in quantities sufficient to adversely affect gold recoveries or operating costs.

Mine production from the Black Fox gold project was shipped to the St. Andrew Goldfields (Stock) mill and the Kinross Gold Macassa mill during the periods April 1997 – September 1999 and October 1999 – May 2001 respectively.

The historical metallurgical performance achieved during the period 1997 to 2001 is summarized in Table 14.1.1.1.

Table 14.1.1.1: Historical Plant Performance

	Actual Reported Production			
Year	**Tonnes 000's**	**Grade gpt-Au**	**oz-Au 000's**	**Gold Recovery, %**
1997	194.5	6.79	39.9	96.38%
1998	308.7	6.67	64.3	96.90%
1999	258.7	5.82	48.3	97.76%
2000	255.2	5.82	46.4	97.04%
2001	81.7	4.81	11.9	98.19%
Totals	**1,098.8**	**5.97**	**210.8**	**97.14%**

Note: Grade reported is the recovered grade, i.e. the grade necessary to produce 210,800oz-Au.

Lakefield conducted comprehensive bench scale testwork in 1996, followed by a combination of pilot plant studies and related bench scale tests in 1999. Metallurgical testwork performed by Lakefield in 1996 demonstrated the Black Fox mineralization to be free-milling and devoid of deleterious elements that could adversely affect the environment or the process. Test results indicated the potential value in deploying a gravity concentration circuit. The program determined the optimum grinds for the West and East Zones to be K_{80} 50µm and K_{80} 30µm respectively. The leach kinetics were found to be most favorable, with 30 hours of leach time being sufficient to achieve optimum results.

The main conclusions developed by the Lakefield work are outlined below:

- The gold mineralization is readily amenable to cyanidation. When grinding in a sodium cyanide solution, approximately 90% of the gold contained in the mill feed is dissolved by the time the pulp has exited the cyclone overflow;
- The degree of dissolution is dependent on the leach feed grind. Optimum size distribution for west zone ore appears to be 50µm while the East Zone mineralization requires grinding 30 to 40µm;
- The Bond Ball Mill work index of the ore varies within the range of 14 to 17kWh/t;
- Gold dissolution is relatively insensitive to variations in leach times over the ranges examined;
- Black Fox mineralization contains no deleterious elements that could adversely affect operating efficiencies or the environment;

- To varying degrees, Black Fox mineralization is amenable to gravity concentration; and
- The ground mineralization exhibits favorable settling characteristics.

Exall entered into a three-year toll milling agreement with St .Andrew Goldfields to process Black Fox mineralization in the Stock mill. The empirical results achieved in the plant confirmed the original test data.

Upon the expiry of the toll milling arrangement with St. Andrew Goldfields, Exall shipped the mine production to the Macassa mill. Metallurgical results continued to confirm the amenability of the Black Fox mineralization to conventional cyanidation followed by CIP technology. While the higher sulfide material generated poorer results and consumed more cyanide, the problems were mitigated through effective blending of the mill feed. Annual gold recoveries exceeded 97% at the Macassa mill.

Programs of laboratory and pilot plant metallurgical studies were implemented in 1999 by Lakefield to examine alternate process options by which overall project economics could be enhanced. Based on examination of six composite samples of varying grades and sulfide content, the use of spiral concentrators was deemed to offer a means by which up to 80% of the gold could be recovered in 15% of the feed weight, given a primary grind of 150µm.

Cyanidation tests, performed on the spiral concentrate, ground to K_{80} 40µm, and un-ground spiral tails, at a nominal K_{80} 150µm, achieved leach residues similar to those achieved in the Stock mill.

Based upon these, and associated test results, preliminary economic analyses were prepared to assess the potential economic consequences of adopting such a circuit. The results of these analyses, conducted by Richard Swider, indicated that the use of a gravity pre-concentration stage, in conjunction with a coarser primary grind, would be worthy of consideration in any future Feasibility Study.

In summary, the Black Fox mineralization is free-milling and environmentally innocuous. Although visible gold is present, relatively fine grinds are required in accordance with the current flowsheet, to achieve optimum results. Upside potential might be realized through the adoption of a gravity circuit in conjunction with a coarser grind. Other process alternatives should also be included in a series of trade-off studies, prior to finalizing the basis for a Feasibility Study. In any event, confirmatory testwork should be performed on samples of mineralization deemed representative of grades and species to be mined in accordance with the new mine plan.

14.1.2 Initial Metallurgical Testwork (1996)

Lakefield Research conducted bench scale test work in 1996 to determine the suitability of the Stock mill to treat mineralization from the Black Fox deposit. The program, designed and directed by Richard Swider, examined recovery of gold by gravity and cyanidation methods. In addition, characterizations of selected samples were performed for environmental purposes.

The work was performed on 67 samples of mineralization, 26 from the East Zone and 39 from the West Zone. The samples were composited into six sample blends, three for each of the two zones, as shown in Table 14.1.2.1.

Table 14.1.2.1: Gold Head Analyses for the Composite Samples, (gpt-Au)

	Calculated[1]	General	Low Grade	High Grade
West Zone				
LR Wtd. Average[2]	9.47	7.71	2.76	25.4
LR with Metallics[3]	8.56	8.37	2.56	22.8
LR Test Average[4]	8.38	6.63	2.18	23.1
East Zone				
LR Wtd. Average[2]	20.1	18.1	3.75	41.0
LR with Metallics[3]	22.9	17.7	9.56	40.0
LR Test Average[4]	19.1	17.6	5.99	35.9

1. Calculated from the weighted average heads from composites General, Low, High.
2. The head grade calculated from the weighted average heads from all samples used for compositing.
3. The direct head for each composite using a ±100 mesh pulp metallics procedure.
4. The back calculated average gold head grade from the test program.

Given the presence of visible gold, albeit finally disseminated, reasonable agreement is achieved in most cases. The East Zone Low Grade demonstrates poor reconciliation between the head values shown.

Detailed head analyses of the individual composites failed to identify any elements or compounds that could be environmentally deleterious or that could seriously adversely affect the cyanidation process. Sulfide sulfur in the East Zone High Grade was measured at 3.05%. The highest equivalent sulfide content in the West Zone material was 0.48%.

The Bond Work Indices of the East Zone and West Zone General Composites were determined to be 16.6 and 14.9kWh/t, respectively.

Gravity concentration tests were performed on each composite sample, yielding the results summarized in Table 14.1.2.2.

Table 14.1.2.2: Summary of Gravity Concentration Test Results

Composite	Head gpt-Au	Wt Recovery, %	Con. Grade gpt-Au	Gold Recovery, %
West Zone				
General	6.63	0.064	5,195	49.9
Low Grade	2.18	0.045	1,531	31.8
High Grade	23.1	0.097	13,132	55.4
East Zone				
General	17.6	0.063	9,580	34.4
Low Grade	5.99	0.069	872	10.1
High Grade	38.0	0.160	15,063	47.6

Lakefield noted that "no coarse (>48 mesh) gold" was observed in any of the gravity tests. Nevertheless, the results suggest that all but the East Zone low-grade mineralization could be amenable to gravity concentration, although free gold commonly observed in the core may be larger.

Cyanidation tests were performed on the gravity circuit tails for each composite to determine the impact that grind and leach time respectively impart on gold dissolution. The results indicated that gold extraction from East Zone mineralization was sensitive to fineness of grind; gold extractions improved as the fineness of grind increased from K_{80} 70µm through 50µm to 30µm. West Zone leach extractions were relatively unaffected by particle size, over the range examined.

In addition, the effects of variable leach times within the range of 36 hours to 72 hours were examined. Gold extractions from both East and West Zone mineralization were found to be insensitive to leach times, again over the range examined.

A summary of the key cyanidation data is presented in Table 14.1.2.3, based on 48-hour leach times, a K_{80} 50µm grind for West Zone material and K_{80} 30µm grind for East Zone mineralization.

Table 14.1.2.3: Summary of Cyanidation Test Data

Composite	Reagent Consumption, kgpt ore		Residue gpt-Au	Gold Extraction, %
	Lime	NaCN[1]		
West Zone				
General	0.70	0.32	0.14	95.1
Low Grade	0.89	0.18	0.08	95.0
High Grade	0.86	0.33	0.12	98.9
East Zone				
General	0.76	0.31	0.68	93.1
Low Grade	1.02	0.20	0.33	93.9
High Grade	0.85	0.52	1.38	92.7

Cyanide consumed during leach, not including initial cyanide to 0.5g/L

The samples examined were very amenable to cyanidation when low dosages of reagents were applied. Grab samples of leach solution taken after 12 hours of leach indicated rapid leach kinetics. Since it was proposed to grind in cyanide solution, Lakefield projected that "a significant proportion of the gold is likely to be recovered in the carbon column circuit".

Gold adsorption test data indicated that no deleterious species were present. It was projected that gold adsorption in a CIP circuit would be rapid and complete after 7.5 hours.

Settling tests were performed on East and West Zone General Composites, at a K_{80} 30µm grind. A favorable unit area rate was achieved, being less than $0.2m^2$/tpd in all cases in which modest flocculant additions were used.

In conclusion, the favorable Lakefield test results, together with the existing Stock mill circuit configuration, supported the concept of milling Black Fox mineralization in the St. Andrew Goldfields' plant.

14.1.3 Stock Mill Operations (1996-1999)

The Stock mill, designed by Leslie Engineering, was constructed in 1988. The plant included the conventional unit processes of:

- Primary crushing;
- Closed circuit, single staged fine crushing;
- Two staged grinding;
- Pre-leach thickener and carbon columns;
- Leach and CIP circuits;
- Carbon stripping and electrowinning; and
- Cyanide destruction.

St. Andrew Goldfields' personnel managed and operated the plant, allowing access to Exall's technical representative. Exall paid a processing charge, based on the tonnage milled. In addition, a bonus was paid to the owner, based on gold extractions achieved.

Black Fox mineralization in excess of 1.6gpt-Au was delivered to the Stock mill by 35t capacity highway trucks. A 610mm x 914mm jaw crusher was replaced with a 1,067mm x 1,371mm unit in 1999. The primary crusher discharge was further crushed in a 1,300mm short head cone crusher, in closed circuit with a screen, prior to being conveyed to the fine ore bin.

Grinding was accomplished in a 2,896mm x 3,658mm, 450kW primary ball mill and a 2,743mm x 3,353mm, 337kW secondary ball mill, in closed circuit with cyclones. Both mills were rubber lined. "Optimum" grinding rates were reported to approximate 43tph, subject to the mineralization being processed, with work indices varying within the range 14 to 16kWh/t. Grinding was performed in cyanide solution.

The cyclone overflow gravitated to an 18.3m dia. thickener, the overflow from which was pumped to seven carbon columns. The thickener underflow was pumped to four leach tanks to provide a nominal 27 hours retention time. Leach tailings were pumped to five CIP tanks that provided 3 hours retention time. Carbon from the columns and CIP circuits was stripped at 142ºC. One tonne batches of carbon were regenerated on-site in an electrically-heated rotary kiln, after washing with 3% nitric acid. Fine "attritted" carbon was recovered and shipped to Noranda. The pregnant strip solution was fed to a $1.0m^3$ electrolytic cell. The sludge produced was dried and charged into an induction furnace to produce doré bars.

Exall's technical representative was present for the monthly estimates of gold inventories and was able to monitor normal operations for about 50% of the time. While operations were satisfactory at the Stock mill, some issues were of concern to Exall. Thus, certain housekeeping issues could have contributed to loss of gold. Further, the Black Fox mineralization was processed in batches, typically of 5,000t.

Thus, at a nominal 1,000tpd milling rate, campaigns were generally of five days duration. Ores from other sources were processed between the Black Fox campaigns, rendering precise metallurgical accounting difficult. This problem was exacerbated by scaling in flowmeters, caused by the high lime additions used in the plant. While the plant produced gold recoveries in the mid- to high nineties, Exall elected to ship mineralization to Macassa upon the expiry of the three year custom milling agreement with St. Andrew Goldfields.

During the milling campaign at the Stock plant, a very short plant test was conducted to operate a Falcon concentrator and gravity table. The initial results were not sufficiently encouraging to justify a protracted test. Based on the results of bench scale and pilot plant test programs, this is somewhat surprising. It is conceivable that insufficient time was available to properly fine tune the circuit. It is also possible that the fineness of grind contributed to the poor results.

14.1.4 Macassa Mill Operations (1999-2001)

Operations at the Macassa plant commenced October 1999. A consulting metallurgist, representing Exall's interests, had free access to all operating information at Macassa and was, in effect, the Chief Metallurgist for the operation's. The plant was highly automated and well-equipped with security cameras. This degree of operations control and the exclusive use of the plant for Black Fox production, mitigated most of the concerns that were associated with St. Andrew Goldfields mill.

The Macassa plant, designed by Wright Engineers, provided the same basic unit processes as those at the Stock plant, although a two-stage fine crushing plant replaced the single stage fine crushing circuit at the Stock mill. In addition, the Macassa plant included a pre-thickener leach tank. Further, being designed to treat 2,000tpd, the plant was oversized for the nominal 1,000tpd Black Fox mine production rate. Accordingly, most of the leach tanks were not required for Exall's purposes. In all respects, the plant was able to satisfy the process requirements.

The crushing circuit comprised a jaw crusher, a standard cone secondary crusher and a tertiary short head crusher, in closed circuit with a vibrating screen. The crushing plant operated 12 hours per day.

Grinding was accomplished in two 600kW ball mills that operated a 24 hours per day, 5 days per week schedule. The primary mill was steel-lined and charged with 100mm grinding media. The secondary rubber-lined mill was charged with 25mm balls. The nominal 1,000tpd (40 to 45tph) milling rate produced a cyclone overflow grind within the range 70% - 75% passing 53 microns.

The cyclone overflow was directed to a pre-leach tank ahead of the 19.8m dia. thickener, the former providing a residence time of 30 hours. Given the favorable leach kinetics, gold dissolution was typically 90% complete by the time pulp entered the thickener.

As for the Stock circuit, the thickener overflow was treated in carbon columns while, for much of the time, the thickener underflow passed through a single leach tank, thus providing 60 hours of leach time in total. The leach tank was only used to satisfy certain logistical requirements, rather than to provide necessary incremental leach time. A 6-stage CIP circuit was deployed, prior to treatment of the carbon in a conventional Adsorption-Desorption-Recovery (ADR) circuit.

Sub-standard metallurgical performance and increased cyanide consumption were observed when treating "sulphidic" material from the S_o zone in August 2000. However, these problems were overcome by blending mineralization prior to milling. Gold recoveries at the Macassa plant are shown in Table 14.1.4.1, together with the measured head grade and calculated head grade.

Table 14.1.4.1: A Summary of Macassa Production Data

Month	Throughput, t ore	Feed Grade, gpt-Au		Au Recovery, %
		Measured*	Calculated**	
October 1999	15,628	7.02	6.03	95.06
November	20,562	9.86	7.66	98.14
December	19,217	8.44	5.11	97.43
January 2000	19,385	11.03	6.05	97.75
February	22,864	12.69	8.59	97.87
March	20,779	3.97	3.56	97.43
April	21,377	4.53	4.16	97.38
May	24,856	7.05	5.16	97.62
June	22,938	10.87	9.85	98.78
July	19,530	8.24	6.26	97.67
August	20,732	4.90	4.53	86.29
September	23,870	4.84	5.12	97.04
October	23,767	6.01	4.79	96.79
November	23,080	6.60	6.78	97.42
December	12,309	7.73	7.00	98.48
January 2001	20,342	6.51	3.85	97.82
February	17,801	7.67	5.88	98.55
March	21,679	4.41	4.55	98.05
April	17,719	4.73	4.09	97.81
May	4,159	3.56	4.23	97.84

*Assay of composite mill feed sample taken by an automatic sample cutter form the mill feed conveyor discharge.
**Calculated head based on in-plant gold inventory, gold production and tailings losses.

Exall reported three plant feed gold contents; the **Measured Head**, the **Calculated Head** grade and the **Calculated Cyclone Overflow** grade.

- The **Measured Head** is the assay of a composite mill feed sample, taken by an automatic sample cutter from the mill feed conveyor discharge;
- The **Calculated Head** is based on the actual amount of refined gold produced, mill circuit gold inventories, the gold contained in the CIP tail residue and solution and the gold associated with the recovered fine carbon; and
- The **Calculated Cyclone Overflow** grade is the computed total assay of this flow, based on solid and solution assays and the pulp density.

Monthly plant gold recoveries were based on the Calculated Head. A review of plant metallurgical accounting procedures indicates the application of sound protocols. The use of the Calculated Head to determine the overall gold recovery was appropriate, given the quality of the raw data used to calculate this head grade. Further, the Calculated Head grade agreed, within reason, with the Calculated Cyclone Overflow gold grade, reflecting the relative ease with which representative samples of the overflow stream could be taken. However, agreements between the Calculated Head and Measured Head values were frequently poor.

Table 14.1.4.2 below compares the Measured Head, Calculated Cyclone Overflow Head and Calculated Head values for the period January through March 2001.

Table 14.1.4.2: Comparison of Head Values (gpt-Au), January to March Inclusive 2001

	Measured	Cyclone Overflow	Calculated
Average	6.28	4.86	5.08
Standard Deviation	5.05	2.01	

The above data is based on 132 shift Measured Head assays and 73 daily Cyclone Overflow calculated values, all reported over the same three-month period. The Measured Head value is most frequently higher than the other two computed values. Further, the variability of the Measured Head assays is considerably greater than experienced with Cyclone Overflow data. The predominant reason for the higher and more variable Measured Head values is probably related to the difficulty in sampling relatively coarse material in the presence of visible gold.

14.1.5 Metallurgical Testwork (1999)

Exall commissioned Lakefield to conduct a program of bench scale and pilot plant tests to investigate the potential for gravity pre-concentration, using spirals and vat leaching as means by which toll milling costs could be reduced. The program was conducted under the direction of Richard Swider. The program was expanded to examine other concepts that offered the potential to enhance process economics.

Samples of high and low-grade mineralization were combined to produce six composites, ranging in grade from 2.07gpt-Au to 14.0gpt-Au. Descriptions of the samples are provided in Table 14.1.5.1.

Table 14.1.5.1: 1999 Test Program Sample Description

Composite	Constituents	Grade gpt-Au
General Composite #1 (GC1)	60% low grade at 5.48gpt-Au 30% high grade at 14.0gpt-Au 10% very low grade at 1.84gpt-Au	10.2
General Composite #2 (GC2)	50% GC1 at 10.2gpt-Au 50% low grade at 5.48gpt-Au	7.67
Low Grade Composite (LG)	100% low grade at 5.48gpt-Au	5.48
Very Low Grade Composite (VLG+A)	75% very low grade at 1.84gpt-Au 25% drum A	2.07
High Grade Composite (HG)	100% high grade at 14.0gpt-Au	14.0
High Sulfide composite (HS+HG)	50% high sulfide at 8.45gpt-Au 50% high grade at 14.0gpt-Au	11.2

The programs involved extensive laboratory and pilot plant work that included gravity concentration, leaching gravity concentrates and tailings, the flotation of gravity circuit tailings, thickening tests and work index determinations. Details of the test programs, and the potential financial implications of the results, are included in the reports issued by Lakefield and Richard Swider.

The conclusions drawn from the 1999 test programs are summarized in point form below.

- The optimum grind for spiral performance was reported to be 166µm;
- At a feed grind of 150µm, the spirals produced a gravity concentrate which, at a 15% weight recovery, contained approximately 80% of the gold in the feed;

- Leach residues approximating those at the Stock mill (0.14gpt-Au) were achieved when the 150µm spiral tailings were subjected to conventional cyanidation;
- Spiral concentrates, reground to 40µm, were leached to produce leach residues of grades (0.13gpt-Au) similar to those at the Stock mill;
- The High Sulfide Composite proved to be more refractory, reflecting experience in the operating plant. Thus, the 40µm leach residue graded 1.8gpt-Au, while the overall leach residue (spiral concentrate and tailings) was in the range of 0.40gpt-Au, equivalent to a 97% gold recovery. Knelson concentrators, used to treat spiral tailings, failed to yield any significant benefit;
- Bond Ball Mill work index determinations indicated a value of 17kWh/t should be used for plant design purposes;
- Flotation was found to be effective in the treatment of sulfide mineralization, but of marginal value when processing low sulfide material; and
- Thickener unit area determinations for the leached spiral tailings (158µm) and the leached spiral concentrate (40µm) were 0.274 and 0.173m^2/tpd respectively, at pH values just in excess of 10.0.

The test programs generated a significant amount of useful information that could be used in process trade-off studies during the preparation of a Feasibility Study. It will be important to ensure that the samples tested in this particular program are representative of the mineralization to be processed in accordance with updated mining plans. Additional confirmatory work might be required for feasibility level work.

14.1.6 Mini Bulk Sample Gravity Tests (2006)

Francis Pitard (2005) recommended 200 mini-gravity tests be completed and compared back to the original sample grades. Apollo started this program with 58 tests completed, averaging about 14kg per test. Of these tests, 47 were completed on “ore grade” material, 6 on “waste”, and five were completed on samples that had not been assayed.

The 47 tests completed on “ore-grade” materials indicated an average gravity recovery of about 59%. The average feed size for these tests was K_{80} 114µm. .

14.1.7 Implications of Test Data and Results on Conceptual Plant Design

The rapid leach kinetics consistently experienced in the laboratory and plant should be exploited to the full. By so doing:

- Gold is extracted at the earliest opportunity;
- The production and handling of high-grade gravity concentrate products is avoided; and
- Leach circuit retention times can be reduced.

Pending further bench scale testwork, a variant of the concept developed in 1999, has been adopted and constitutes Owner Mill Case 1 in this Preliminary Feasibility Study. Thus, the primary grind will be 80% 150µm and a gravity concentrate will be produced from the cyclone overflow product. The concentrate will be reground to 80% 40µm, prior to leaching. However, in this case, the reground concentrate will be combined with the gravity tailing to feed the pre-leach thickener. By using this circuit, and grinding in cyanide, the recovery of gold to the gravity concentrate will be reduced significantly. Nevertheless, such a circuit will enhance the rate at which minor amounts of the more refractory mineralization will be leached. Further, since testwork has demonstrated gravity recoveries improve with increasing feed grades, the proposed circuit will alleviate any potential difficulties that may result from short-term feed grade "spikes". Further gravity and leach work, in conjunction with mineralogical and modal analysis, is required to examine the circuit, at a bench scale level, prior to the preparation of a detailed Feasibility Study.

Historically, leach times of 30 hours have been adopted. Given the operating and test results reported above, a leach time of 18 hours should provide adequate time to achieve the optimum economic leach. This supposition must be confirmed in trade-off studies based on the laboratory work noted above.

14.1.8 Gold Recovery Projections

Gold recoveries are plotted against feed grades in Figure 14-1 for the period January 2000 through to the termination of operations in May 2001. With the exception of August 2000, all gold recoveries exceeded 96%, indicating a modest improvement in gold recovery as feed grades increased from a nominal 2.5gpt-Au to 10.0gpt-Au.

The precise reason(s) for the poor gold recovery in August 2000 were never clearly identified. It was strongly suspected, however, that the milling of S1 mineralization, reportedly higher in sulphides content, could have contributed to the problem. Exall determined that swings in gold recovery could be largely mitigated by blending ores prior to treatment.

An overall gold recovery of 96% is used for the pre-Feasibility Study, based on the assumptions that:

- Ores to be treated will exhibit similar metallurgical characteristics to those processed in 2000 and 2001;
- The feed grade will not fall beneath 4.0gpt-Au, and
- Ores of differing grades and mineralogical composition will be blended, at least to the same degree as that achieved during the Macassa operations.

SE has determined that the overall gold recovery for the purposes of this report will be 95%. This conservative value, which could be increased based on further historical data review, presumes lower recoveries from east zone ores. Such low recoveries from this ore were observed in lab studies and actual Macassa operations. Macassa suffered the low recoveries when unanticipated high sulfide ore was processed. When operations were forewarned of the high sulfide ore, it was blended with low sulfide ore and recovery remained high.

14.2 Stock Mill

The Stock mill, operated by Apollo Gold Corporation, has the capability of a throughput of 1,100tpd (396,000tpy) on ore to be processed from the Black Fox mineralization. Metsim computer simulations of the grinding circuit have determined that a series circuit, utilizing an existing third grinding mill along with reconfigured existing cyclones, will produce the optimal grind size at the rate necessary to achieve 1,100tpd. The recovery projections for the Stock mill are 95%, which is considered achievable at the given grind size of P_{80} = 55µm and leach times in excess of 24 hours.

The Stock Mill is located 33km west of the Black Fox Project. The mill was built in 1988 and was designed by Leslie Engineering of Toronto. The cost in 1988 was US$17million.

It was initially designed to process 500tpd through a conventional fine grinding, cyanide leach gold recovery plant. The plant was expanded in 1993 to 800tpd and additional equipment was added in 1997 to increase the capacity to a maximum of 1,300tpd depending on the fineness of grind required. The mill was designed to treat ore from the Stock Mine whose head frame is adjacent to the mill. The design also provides for treatment of custom ore delivered by highway trucks to the large storage pad adjacent to the crushing plant. The Stock mill flowsheet from 1998 provided by St. Andrew Goldfields Ltd. is shown in Figure 14-2.

14.2.1 Process Description

Process flowsheets have been prepared to the degree of detail deemed appropriate for this Feasibility Study. The following plant description outlines the process facilities flowsheet and layout.

Crushing

From a stockpile on the storage pad ore is reclaimed by front end loader into a feed hopper equipped with a vibrating pan feeder that discharges on to a 1067mm wide conveyor belt. This conveyor discharges on to another 1067mm wide conveyor that feeds a 610mm x 914mm Kemco jaw crusher. The jaw crusher product falls on to a 610mm wide conveyor belt where it is joined by the product of a Metso HP 300 cone crusher. This belt discharges on to another 610mm wide belt that feeds a 1,829mm x 6,096mm Metso TS 302 double deck vibrating screen. The oversize from both decks falls into the cone crusher and the undersize falls on to a 610mm wide conveyor belt that discharges into a cylindrical fine ore bin. The crusher building is equipped with a baghouse dust collector.

Grinding

Crushed ore is fed from the fine ore bin by two 1067mm wide belt feeders onto a 610mm wide conveyor that discharges into the feed chute of a 2.9m x 3.7m, 450kW (600hp) primary ball mill. The mill is equipped with rubber liners and is charged with 70mm grinding balls. This mill discharges into a pump box from which it is pumped to a 305mm diameter cyclone and a 381mm diameter cyclone operated in parallel. The cyclone underflows falls into a splitter box where they can be directed back to the primary mill or to the 2.7m x 3.4m 337kW (450hp) regrind mill. The cyclone overflows flow to the regrind mill discharge mill pump box. The main regrind mill is equipped with rubber liners and it is fed with a portion of the primary ball mill cyclone underflow and a portion of the underflow from three 254mm diameter secondary cyclones.

The main regrind mill discharges into a secondary cyclone feed pump box where it is joined by the overflow from the primary cyclones and the discharge of the second regrind mill. The secondary cyclone underflows fall into a splitter box where they can be sent to either regrind mill. The second regrind mill is a 2.3m x 2.4m, 225kW (300hp) ball mill also equipped with rubber liners. The discharge from this mill is pumped back to the secondary cyclone feed pump box. The overflows from the secondary cyclones flow to a 914mm x 2438mm vibrating trash screen. The trash screen undersize falls into the thickener feed pump box. Milk of lime and sodium cyanide solution are added to the primary mill feed chute and most of the contained gold is in solution by the time the ore reports to the secondary cyclone overflow.

Leaching

The ground ore slurry is thickened in an 18.3m diameter x 3.7m high Eimco thickener. The thickener overflow is pumped to two trains of carbon in column (CIC) tanks operating in parallel where the gold is adsorbed onto 6–12 mesh granular carbon.

One train of CIC tanks is made up of four 0.9m diameter x 3m high columns and the other of three 1.5m diameter x 3.6m high columns. The CIC tails is pumped to either the mill solution tank or directly to the primary grinding circuit. The loaded carbon goes to a 1,219mm diameter Sweco loaded carbon screen and to the loaded carbon tank. The thickener underflow is pumped to a series of four 7.6m diameter x 7.6m high agitated leach tanks arranged for gravity flow between tanks through 203mm diameter pipes. The overflow from the last leach tank flows to a series of five CIP tanks where the gold in solution is adsorbed onto 6-12 mesh granular carbon. Carbon is advanced between stages and to a Sweco 1,219mm diameter loaded carbon screen by pumps in the CIP tanks. The loaded carbon from this screen is also pumped to the loaded carbon tank. The tails from the last CIP tank passes through a 914mm x 2438mm carbon safety screen to a pump box that used to be the start of a two-stage cycloning circuit for producing underground mine backfill. The backfill cyclones have been removed and the existing sumps and pumps are now used to send the tailings slurry to the tailings pond. The carbon fines from the carbon safety screen are dewatered in one cubic meter boxes and sent to a smelter.

Carbon Stripping

From the loaded carbon tank the carbon is transferred to a 1t per batch strip tank where the gold is removed from the carbon by a concentrated solution of caustic and sodium cyanide at 142°C. The pregnant solution flows out of the strip tank through a heat exchanger to an electrowinning cell where the gold is plated onto stainless steel electrodes. Periodically the gold sludge is removed from the electrodes by pressure washing. It is then pumped out of the electrowinning cell by a small diaphragm pump into a small filter press where it is dewatered. Periodically the sludge is removed from the filter press, dried and smelted in an induction furnace before being poured to produce a doré bar. The tails from the electrowinning cell are pumped into a barren solution tank. From there it is pumped through the heat exchanger and an electric inline heater back into the strip vessel. Stripped carbon is transferred from the strip vessel to an acid wash tank where scaling is removed by a 3% nitric acid wash before the carbon is sent through a carbon regeneration kiln. The kiln is operated at 700°C to remove volatile contaminants from the carbon.

The discharge from the kiln falls into a quench tank and from there it is transferred into a carbon conditioning tank. New carbon is also added to this tank and, after attrition agitation, it is transferred to a 1,219mm diameter Sweco screen where fine carbon is removed. The oversize from that screen falls into the reactivated carbon tank from which it is transferred to the CIC or CIP tanks as required.

Water Management

Makeup water is pumped from the tailings pond by a barge-mounted pump either to the mill solution tank or into the first of two water treatment tanks. Sodium metabisulfite and copper sulfate solutions are added to the first tank to destroy the residual cyanide and ferric sulfate is added to the second tank to precipitate arsenic. From the second tank the treated water flows into a polishing pond to allow for completion of the cyanide destruction reactions and settling of arsenic particulates before it is discharged into the environment.

14.2.2 Historical Processing of Black Fox Ores

Ore from the Black Fox Mine property was custom milled at the Stock mill from April 1997 to September of 1999 operated by St. Andrew Goldfields. The ore was free milling and finely disseminated so cyanidation of the whole ore was undertaken, rather than gravity pre-concentration. Despite visible gold (vg) in some geological samples no coarse gold (>48 mesh) was noted in any of the subsequent Lakefield gravity tests.

Ore was ground to a P75 (75% passing) size of 56µm in a conventional CIP mill, having 40 hours retention time at 800tpd, or 30 at 1,100tpd. Mine production was generally 900tpd of mill grade ore, unless development was "in ore", which increased output. Run of mine ore was graded between "high grade ore" sent to the mill, and "low grade ore" assaying under 1.6gpt, which was below the cut-off grade for economical milling (including transport). This low grade ore was used as foundation for ore piles on unsealed ground, and to purge the mill prior to shutdowns. Ore was transported in 35t road trucks and weighed over an automated load-bridge at the receiving mill. Prior to the addition of a 42in x 54in Eagle Crusher in 1999 the original 24in x 36in 60hp Kemco jaw crusher had been undersized putting a heavy (50mm: 2in feed) load on the 4.25ft (200 hp) Symons SH cone, resulting in frequent maintenance.

Excluding a retrofitted tertiary grinding mill that had surge problems, the Stock mill facility had a 450kW (600hp: Allis Chalmers 9.5Øx12ft) primary mill and 337kW (450hp: 9Ø x 11ft) secondary mill, both rubber lined. Optimum grinding was achieved at about 43tph depending on ore type, which had a bond index between 14 – 16kWh/t.

Ore was ground in cyanide to minimize gold entrapment within the mills, and then thickened in a 60ft (18.3m) diameter Eimco thickener, leached (4-stage) and mixed with 6-12 mesh granular carbon in 5 (five) 4.27Ø x 4.27 m (14Ø x 14ft) CIP tanks. The thickener overflow was also contacted with carbon in 4 carbon columns (expanded to 7, of which 5 were generally on line in 1998). Gold recovery was via a 1m³ (35.5ft³) electrolytic cell and induction furnace. Tailings disposal was the responsibility of the custom miller. Check assays were conducted at several commercial assay laboratories, normally on random feed grade samples.

Carbon was separated from the pulp using basket screens, and a vibrating (20#) screen. One tonne batches were regenerated on site in a 700°C electrically heated kiln after washing the carbon with 3% nitric acid. Loaded carbon was stripped at 142°C (290°F). Fine, attrited, carbon was decanted and drained into wooden boxes (1m³).

Entrapped gold was recovered by incinerating this product at Noranda Smelter complex in Quebec, which recovered 92% of the contained gold (less minor material penalties and handling costs).

Doré bars were poured under the joint supervision of the custom miller and miner, and shipped (via Brink's) to Johnson Matthey (Canada) in Brampton ON, for refining (95% credit on contained gold). The doré typically contained 5.5% of Silver (range 2.8 – 7.6%), which was paid in full at current metal price. However, no special efforts were made to recover silver in the circuit. (Silver in the ore was not routinely assayed for [generally it was under 5gpt], and no measure of silver recovery was conducted, though some higher silver grades were detected).

Liquid sodium cyanide (delivered in bulk tankers) by Degussa was used along with Calgon GRC#22 granular carbon.

A gravity circuit consisting of a Falcon concentrator and vibrating table were installed soon after start-up but after one week the concentrate in the bowl contained 2,000gpt material, which was significantly less than the 59,000gpt found behind the mill liners, and further testing was aborted (Exall project report June 1997).

14.2.3 Projected Processing of Black Fox Ores

The Stock mill ran Black Fox ore from April of 1997 to September of 1999 at an average throughput of 900tpd with a recovery of above 96%. Though the operational procedures were adjusted to accommodate the specific ore needs, several opportunities exist that could potentially improve the throughput to 1,100tpd while maintaining the high recoveries formerly achieved. Based on Mestim modeling and historical data SE believes that, with the appropriate modifications, the Stock mill, when operated by Apollo Gold, will run at an average daily throughput of 1,100tpd or 396,000tpy with an average recovery of 95%.

A 1,100tpd operating throughput will translate to 53tph after an 87% availability factor is applied. 5% lower availability has been applied to the Stock mill operation due to the age of the mill and the operational difficulties that could arise from alterations from its original design parameters. Metsim modeling of the existing circuit, taking advantage of the third (300hp) grinding mill, indicates that 53tph is achievable with minimal modifications to the cyclone configuration and operating parameters.

Crushing

During the Black Fox ore campaign, it was determined that crushing limited the capabilities of the grinding circuit to produce the size of material required to obtain optimal leach kinetics. Improvements in throughput where observed when the crusher was upgraded from the Kemco (24in x 36in) to the Eagle (42in x 54in) jaw crusher. The current jaw crusher at the Stock mill is again the Kemco jaw crusher. The former (HP200) cone crusher was replaced with a (HP300) cone crusher which is capable of producing P_{80}=10mm material at a rate of up to 140tph.

Since ore will be crushed on site before hauling to the Stock mill, the effect is a three stage crushing circuit that will accommodate the necessary feed size to the primary mill for such operations.

Grinding

During the operation of the Stock mill on the Black Fox ore only two of the three available mills were used on a consistent basis. Two circuits were examined in Mestim: the first with the third mill working in parallel with the other secondary mill; and the second option with primary, secondary, and fine (tertiary) grinding, each in closed circuit with dedicated cyclones. Either option is achievable with minimal capital investment.

Metsim modeling shows that using the third mill in the series configuration and converting the primary mill to steel liners will have a significant effect on the grinding circuit production. It is projected that a better grind can be achieved with higher throughput (from 75% passing 270 mesh [nominal 55µm] at 900tph to 80% passing 270 mesh at 1,100tpd). A parallel secondary circuit, while a simpler modification to implement, has proven to be more difficult to model at the same throughput due to the differences in the parameters of the parallel secondary mills operating in closed circuit with a single cyclone configuration. This configuration, using the same availability assumption and a steel lined primary grind, was limited to 1,000tpd operating rate to achieve the same grind.

Leaching

Leaching studies indicate that Black Fox ore is extremely amenable to cyanidation and exhibits good leach kinetics, requiring at a maximum, 24 hours leach time. The Stock mill's leach residence time is 30 hours at 1,100tph. This is more than enough time to leach the Black Fox ore. The Black Fox processing facility design includes a pre-leach tank to take advantage of the rapid kinetics and send more gold to the CIC as was proclaimed to be effective in the Macassa operations. The opportunity exists to reconfigure the leach circuit to provide for a pre-leach tank.

Another, simpler, option would be to convert the last leach tank to a CIL tank and begin the adsorption process sooner. This would improve gold loading which may become an issue in the one-tonne carbon stripping circuit.

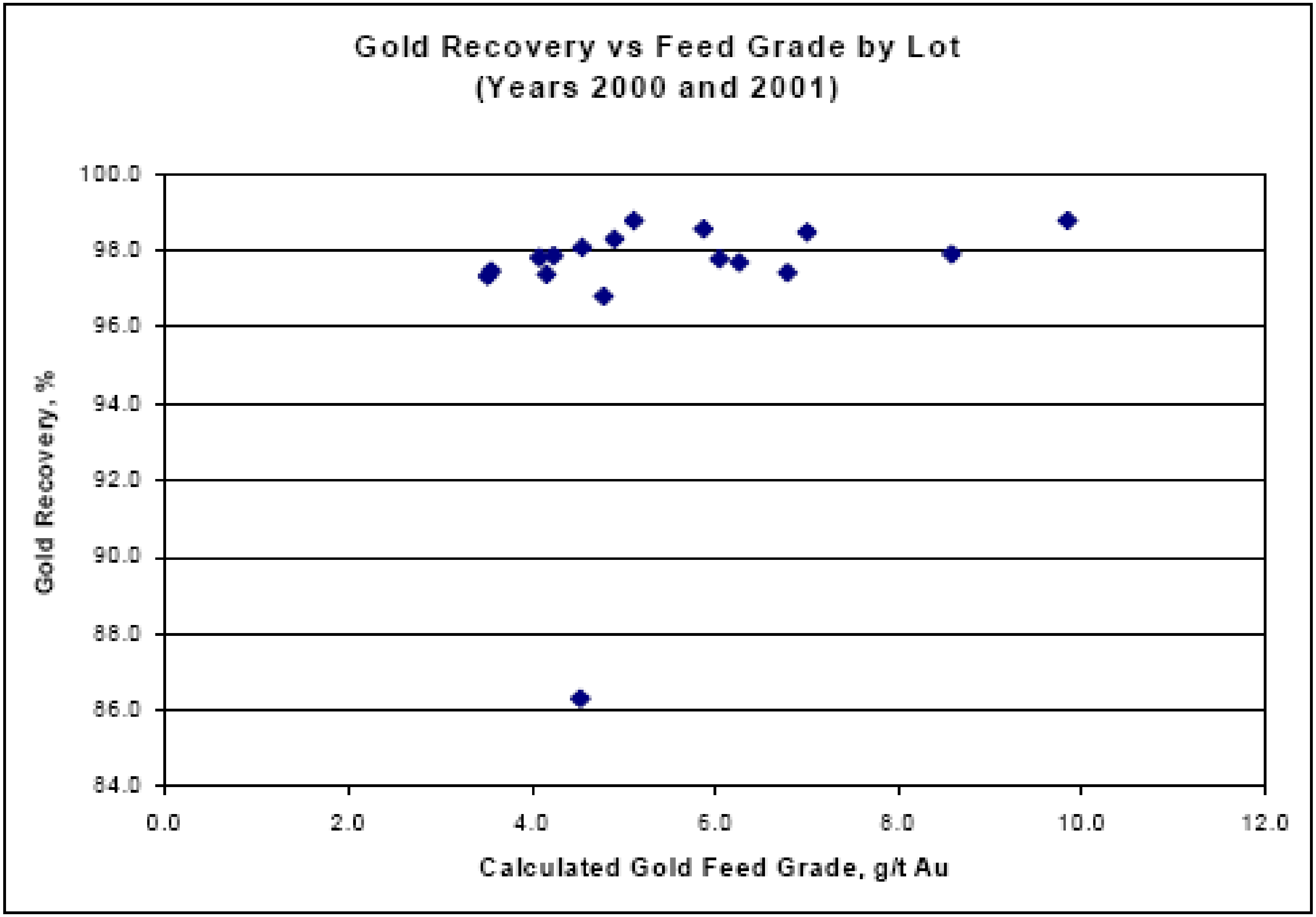
Carbon Stripping

The Stock mill is equipped with a one tonne carbon circuit, capable of averaging approximately 1.5tpd of carbon stripping under optimal conditions.

At 95% recovery and an average head grade of 6.5gpt, carbon will be loaded at a nominal 4500gpt for 1.5t per stripping or 6,800gpt running a single strip per day. These loading levels are higher than is generally expected of carbon in slurry applications, but are achievable. Carbon loading is improved with regeneration; and at 1.5tpd stripping, the 1t regeneration kiln will have to be bypassed by one third of the carbon going through the strip circuit. A thorough review of this circuit is recommended.

Water Management

The current water management system is expected to continue operation as it currently exists.



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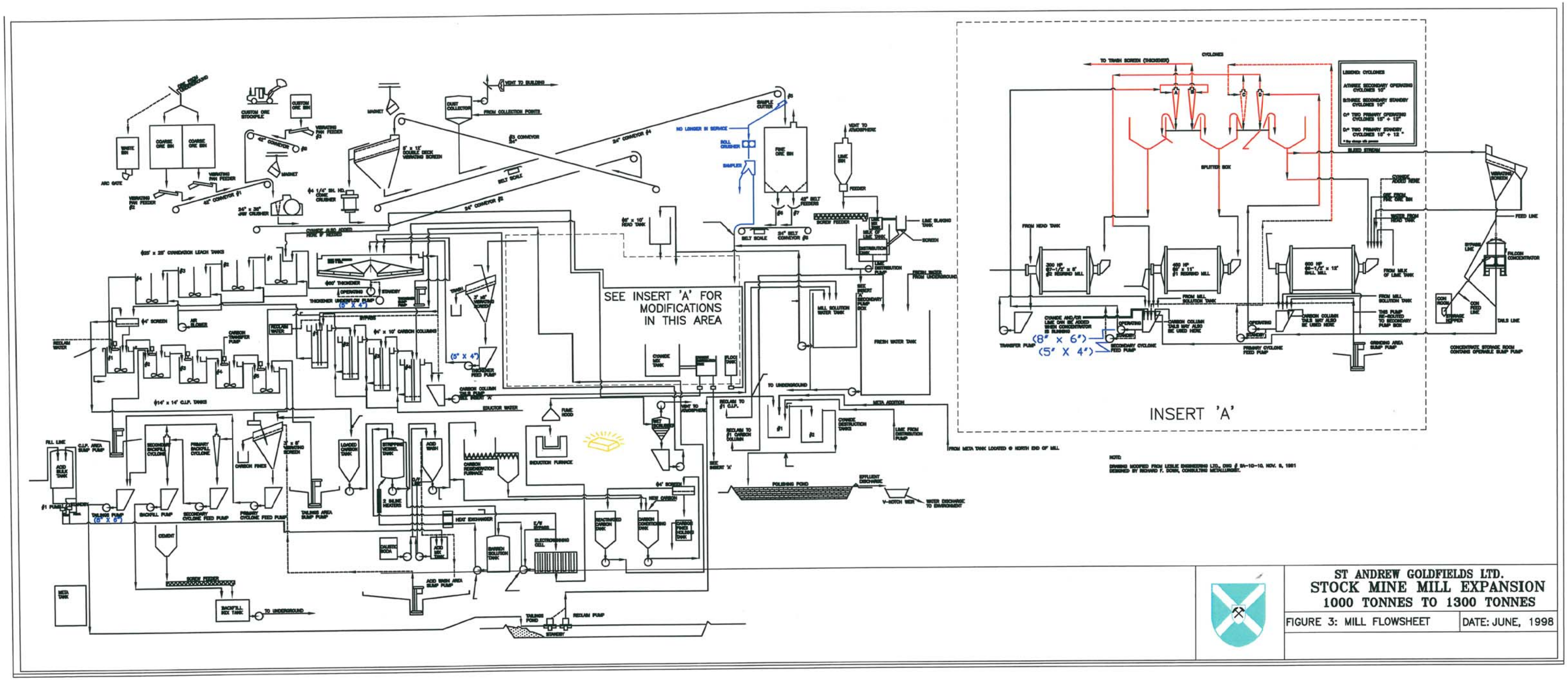
File Name: Figure 14-1.doc

Black Fox,
Timmins, Ontario, Canada

Source: Mine Development Associates

Gold Recoveries vs. Feed Grades

Date: 08-07-07 | Approved: DKY | Figure: 14-1



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ApolloGold

TIMMINS, ONTARIO, CANADA
SOURCE: ST. ANDREW GOLDFIELDS, LTD.

BLACK FOX

STOCK MILL PROCESS FLOWSHEET

DATE:	APPROVED:	FIGURE:	REVISION NO.
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15 Mineral Resource and Mineral Reserve Estimates (Item 19)

15.1 Drillhole Database

The drillhole sample database was compiled by Apollo, reviewed for QA/QC by Analytical Solutions of Toronto Canada and is determined to be of good quality. The database, consists of five Microsoft Excel spreadsheets containing collar locations, drillhole orientations with down hole deviation surveys, assay intervals with results, geologic logs and geotechnical logs. SRK has reviewed previous data verification work, the procedures used by Apollo to update the database and has conducted spot checks of its own finding no problems or mistakes. The assay database contains several columns of gold assays representing the original 15g and 30g assays as well as numerous repeat assays including some screen metallic analyses. The Au assay data used for this Resource estimate was the original assay value unless repeat check analyses had been made. In this case, the average of all the assay results were used in place of the original assay.

The Resource database contains information from 1,889 drillholes totaling 335,983m of drilling. The maximum drillhole depth is 995m and the average is 178m. Most of the holes were drilled inclined to the north in order to intercept the south dipping mineralization at a high angle. Drillholes have been collared both on surface and underground.

Analysis of the sample intervals shows that the majority range between 0.5m to 2.0m however, there is a very small percentage of intervals up to 8m. The average sample interval is 0.9m.

15.2 Block Model

The Black Fox deposit was modeled only for gold content. The model has a uniform block size of 3m x 3m x 3m. All block estimates were made using only the 1.5m drillhole composites. The model boundaries based on local mine grid coordinates are presented in Table 15.2.1.

Table 15.2.1: Black Fox Model Limits

Direction	Minimum	Maximum
Easting	9,700	10,651
Northing	9,500	10,352
Elevation	9,400	10,000

15.3 Geologic Model

The Black Fox deposit is described by Prenn (2006) as follows:

"*Gold mineralization at the Black Fox deposit occurs in several different geological environments within the main ankerite alteration zone, which has an indicated strike length of over 1000 m and a variable true width ranging from 20 to over 100m . This mineralized envelope occurs primarily within komatiitic ultramafics and lesser mafic volcanics within the outer boundaries of the Dester-Porcupine Fault Zone. The auriferous zones have several modes of occurrence, from concordant zones which follow lithological contacts and which have been subsequently deformed, to slightly discordant ones which are associated with syenitic sills and quartz veins or stockworks.*"

For this study, the mineralization is subdivided into three main domains based on the continuity and style of the mineralization. The first is called the "Main Zone" and is delineated by the primary domain of shearing and mineralization. It is broader near surface reaching a maximum true width of 150m normal to strike and dip and narrows at depth. It averages approximately 80m normal to strike and dip and has currently been drill tested to 600m below surface. Within the "Main Zone", the mineralization occurs along both a foliated fabric cut by discrete shear zones and as stockwork carbonate veining. The second mineralization domain is called the "Flow Zones". This mineralization occurs as numerous sigmoid and lens shaped bodies completely hosted within or adjacent to the "Main Zone". This gold mineralization within these bodies has good geologic and grade continuity. The rock is distinctive with strong foliation, pervasive shearing and can be correlated reasonably well between adjacent drillholes. The third mineralization domain is called the "Hanging Wall Zone". This includes all mineralization within the hanging wall above the "Main Zone" and a small portion in the footwall. Within this zone, the gold mineralization consist of small discontinuous pods localized along discrete structures. Each of the three mineralization domains were modeled independently.

15.4 Basic Statistics and Compositing

The gold assay data was first plotted on histogram and cumulative frequency graphs to understand the basic statistical distribution of the raw data. The histogram plots show a strong positive skewness and the cumulative frequency plot illustrates a continuous population set with no major changes in slope within the main data population. The cumulative frequency plot does show several outlier data values at the upper end of the grade distribution.

The raw drill data was composited into 1.5m intervals starting at the collar and continuing to the bottom of the hole. The appropriate codes for missing samples and no recovery were used during the compositing procedures. The 1.5m assay composites were also plotted on histogram and cumulative frequency graphs for comparison to the raw data and to access appropriate capping levels. Review of the cumulative distribution plot with composites greater than 100gpt showed a fairly continuous distribution of Au values up to about the 275gpt level (Figure 15-1). Based on the outlier nature of the composites above this level and statistical validations of models run at a variety of capping levels, the Au values were capped at 250gpt after compositing, to prevent the over estimation of grade in the block model. This resulted in 24 drillhole composites ranging from 252gpt to 1,169gpt being reduced to 250gpt-Au.

15.5 Specific Gravity

Specific gravity testing has been carried out, in house, by Apollo and described by Prenn (2006) as follows:

"*A total of 1,218 density tests have been completed by Apollo from core intervals. The average density of mineralized material is 2.78", while the average density of unmineralized material is 2.85."* Apollo was skeptical of these results and further refined this data by sending 107 samples to an outside laboratory for independent analysis. The laboratory results reported an average density of 2.84g/cm for the ore. This value was used for all material in this Resource estimate.

15.6 Variogram Analysis

Variogram analysis was conducted on the 1.5m drillhole data to determine appropriate projection ranges and to test for any preferred orientation of the mineralization. The composites were first flagged to differentiate them into data sets to be used for an indicator estimation technique. All composites greater than 0.5gpt-Au were flagged as “mineralized group” and those below this cut-off were flagged as “non-mineralized group”. Variograms were then constructed using Vulcan software along all directions within the plane of the mineralization and perpendicular to it using only the ore group composites. The greatest range was oriented within the plane of the mineralization, striking at azimuth 73º dipping -50º S. The shortest range is oriented perpendicular to the plane of mineralization at azimuth 343º dipping -40º. Ranges, nugget values and total sill values are presented in Table 15.6.1.

Table 15.6.1: Variogram Results for 1.5m Composite Data

Orientation	Range	Nugget	C_1 Sill Differential
All directions within plane 73°,-50°	30m	110	55
343°, -40°	20m	45	80
Model Variogram		110	55

15.7 Grade Interpolation

Geologic hard boundaries were used to confine the modeling of the “Main Zone”, the “Flow Zones” and the “Hanging Wall Zone”. For this purpose, drill log data was used to differentiate the characteristics of each zone in order to create 3-D solid shapes. The “Main Zone” is composed of a single body along the entire strike and dip of the current drilling. The “Flow Zones” consist of ten distinct bodies, all of which are located within or along the foot wall of the “Main Zone”. The “Hanging Wall Zone” includes all mineralized intercepts in the hanging wall rocks above the “Main Zone” and a few in the footwall. These three domains were all treated as hard boundaries such that the blocks within them were estimated only from composites also located within them. Two different estimation techniques were used within the three domains.

The grade estimation of the “Main Zone” and “Hanging Wall Zone” was conducted using a categorical indicator approach due to the strong spatial grade variability seen in the drilling data. This method first separates the composite data into lower grade and higher grade groups based on an appropriate cut-off value, in this case 0.5gpt-Au. Indicator values are then flagged into the composite data such that values below 0.5gpt-Au are flagged with a “0” and those above are flagged with a “1”. These composite indicator values are then interpolated into the block model thus creating indicator block values between 0 and 1. The indicator values were interpolated using an Inverse Distance Cubed algorithm. A min/max of 1/3 composites with an octant maximum of 2 was used. The estimation was allowed to search within a 35m x 35m x 9m ellipsoid, oriented within the mineralization plane striking 073° and dipping -50° south. The interpolation criteria were derived mainly from numerous trials and visual inspection of the results.

This procedure effectively assigns a probability to each block as to whether it would be above the 0.5gpt-Au cut-off. Blocks assigned with a value of 0.1 have a 10% probability, a 0.5 have 50% probability and those with a 1.0 have a 100% probability. For this model, all blocks assigned with a value of 0.2 or 20% Probability and above, are considered to be within the 0.5gpt-Au grade shell. The composites are next flagged with the interpolated block indicator values so that all composites located within the grade shell can be selected during the grade interpolation. The flagging procedure forces the grade interpolation to use all composites located within the indicator blocks regardless of there grade. The categorical indicator technique precludes the necessity of creating very complex wire frame grade shells to control grade assignment.

The “Flow Zones” were modeled using wire frame hard boundaries as confining grade shells. Each of the ten individual bodies were combined into a single triangulation and all blocks located within this triangulation were allowed to receive a grade estimation.

The next step was to run Au grade interpolations within the three mineralization domains. Blocks within each zone were estimated using only composite data from within that same zone. Numerous grade estimations were conducted using several different inverse distance weighting powers and a variety of different modeling criteria. The results of each estimation, were analyzed by point validation. This procedure removes a drillhole from the database and then estimates grade for all the composites within it. It then removes the next drillhole etc until all composites have been estimated. An x-y scatter plot is then constructed from the estimated versus actual grades to evaluate the correlation. Numerous point validation runs are made to test the effects of different min/max composites/blocks, octant search limitations and minimum number of drillholes required to assign grade. Once an optimal set of estimation parameters is determined by point validation, the actual grade interpolation is then conducted. The Au grade interpolation was then analyzed within the historical underground mining area for reconciliation against the reported production data and the block model grades are compared statistically to the composites used to estimate them. These results are used to adjust the various modeling parameters until an appropriate estimation was achieved.

For the grade assignment, the three zones were estimated by differing Inverse Distance powers. The “Flow Zones” which represents strong, relatively continuous mineralization, was estimated using an Inverse Distance Cubed algorithm. The “Main Zone” and “Hanging Wall Zone” which represent less continuous mineralization were estimated using an Inverse Distance Squared algorithm. All these estimations required a minimum of three and a maximum of five composites to assign grade to each block. In addition, an octant search restriction was applied allowing a maximum of two composites from each octant. No restriction was placed on the number of drillholes required to assign grade. For the “Main” and “Hanging Wall Zones”, the composites flagged within the indicator blocks were combined with the indicator composites outside of the blocks and both data sets were used to assign grade. This combined data set was used so that composites which were above the indicator cut-off and close to the blocks could be used for the grade estimation. A search ellipsoid with a range of 35m down dip, 35m along strike and 9m across strike and dip was used. These ranges are based somewhat on the variography but mainly on the author’s evaluation of what is appropriate for the deposit. The low number of composites per block and the short search ranges were purposely chosen to restrict the grade assignment to consider only the drillholes in close proximity to the blocks. These techniques were chosen to try to emulate a pod like nature to the mineralization and prevent grade dilution into excessive blocks. The average distance to all composites and the number of drillholes used to estimate grade in each block were stored for later use in Resource classification.

15.8 Block Model Validation

The Black Fox model was validated using three procedures. First, the interpolated block grades were visually checked on sections and bench plans for comparison to the composite assay grades. Second, grade reconciliation against the historical underground production was evaluated. Third, statistical comparisons were made between the interpolated block data, composite data and raw assay data.

The Black Fox deposit historically produced 1.1Mt of ore with an average grade of 6.03gpt-Au. Apollo has good survey records of all the underground development and has created 3-D solid shapes of these workings. To verify the SRK model, the block model was "mined" within the 3-D solid of the historical workings to determine the percentage of each block included within it. All full and partial blocks within the mined solid were tabulated for total tonnes and average grade for comparison to historical tonnes and grade of production. The SRK model was specifically designed to approximate the historical production while remaining conservative. Table 15.8.1 presents the results of this analysis.

Table 15.8.1: Model Validation by Comparison with Historical Production

Model	Au Cut-off	Mt	Grade Au gpt	Moz	Estimated vs. Mined (oz)
Historical Mine	Unknown	1.1	5.97	0.210	
SRK	1gpt	0.538	9.7	0.167	79%
	3gpt	0.319	15.00	0.154	73%

The final model validation is based on statistical comparisons between the interpolated block grades, 1.5m composite grades and raw assay grades for individual model zones and the entire model. This included evaluation of histogram patterns, cumulative distribution plots and basic statistical values (Table 15.8.2).

Table 15.8.2: Statistical Comparisons of Au gpt within Raw Assays, Composite Assays and Block Model Assays

Model Domain	Data Group	Mean	Median	Maximum	Variance	No. of Samples
Flow Zone	Raw Assays	7.4	0.7	1942	2016	2717
	1.5m Composites	4.9	0.8	250	207	1690
	Blocks	4.5	1.8	178	67	28,821
Main Zone	Raw Assays	4.5	0.5	3884	2274	26,669
	1.5m Composites	3.9	0.8	250	247	14,219
	Blocks	3.9	1.4	250	90	223,257
Hanging Wall Zone	Raw Assays	1.7	0.6	250	67	2621
	1.5m Composites	1.5	0.6	216	48	1,400
	Blocks	1.4	0.7	127	27	92,548
All Zones	Raw Assays	4.7	0.6	3884	2150	30,841
	1.5m Composites	3.8	0.8	250	227	17,309
	Blocks	3.3	1.1	250	72	344,626

Previous studies conducted by SRK have shown that the Black Fox deposit is not sensitive to modeling algorithms. During the Pre Feasibility Study, Ordinary Kriging, Inverse Distance Squared and Inverse Distance Cubed techniques were used to model the deposit. The various techniques display minor statistical variations but they all produce very similar average grades and tonnes. For this reason, this exercise was not repeated here. A typical block model cross-section of Au grade with Au composite data shown in Figure 15-2.

15.9 Adequacy of Resource Estimation Methods

The Black Fox deposit has been estimated using a modern block modeling technique. This included proper geologic input, appropriate block model cell size, assay compositing and reasonable interpolation parameters. The results have been validated using three methods including; on screen proofing, rectification to historic production and statistical comparisons between the estimated block grades and the composites used to assign them.

15.10 Mineral Resource Classifications and Resource Statement

15.10.1 Resource Classification

The Mineral Resources are classified under the categories of Measured, Indicated and Inferred Mineral Resources according to CIM guidelines. Classification of the Resources reflects the relative confidence of the grade estimates, as a function of many factors including primarily; assay data quality, QA/QC procedures, quality of density data, and sample spacing relative to geological and geo-statistical observations regarding the continuity of mineralization.

In this study, the blocks were assigned to Indicated or Inferred based on the average distance to the composites and the number of drillholes used to estimate grade. The blocks estimated by at least two drillholes from at least two composites, where the distance to the closest composites was within 15m or less were classified as Indicated. The confidence of the projection distance used to determine Indicated Resource was determined from the drillhole variograms, reflecting a distance of approximately one-half the average range and the necessity of more than one drillhole and composite to predict the Au grade.

The blocks located greater than 15m from the nearest composite were classified as Inferred. Thus the Inferred category does contain blocks (approximately 10%) estimated by a single drillhole, by a single composite; some of which are located up to 35m away. The material estimated by a single drillhole and a single composite was isolated and compared statistically to that estimated by multiple drillholes. This showed a slightly lower grade for the blocks with the lower confidence and was therefore considered defensible for assignment to the Inferred category.

15.10.2 Mineral Resource Statement

The tonnage and grade for Indicated and Inferred Resources differentiated by different mining methods at appropriate Au cut-offs are shown in tables 15.10.2.1 and 15.10.2.2.

Table 15.10.2.1: Black Fox Indicated Resource Statement

Mining Method*	Category	Cut-off gpt-Au	Mt	Grade gpt-Au	Contained oz-Au (000's)
Open Pit	Indicated	1.0	4.8	5.3	813.1
Underground	Indicated	3.0	1.7	11.4	622.6

* Mining Method is determined by relative location above or below the 9,814.5m elevation

Table 15.10.2.2: Black Fox Inferred Resource Statement

Mining Method*	Category	Cut-off gpt-Au	Tonnes	Grade gpt-Au	Contained oz-Au (000's)
Open Pit	Inferred	1.0	2.7	4.7	408.3
Underground	Inferred	3.0	0.8	13.1	329.0

* Mining Method is determined by relative location above or below the 9,814.5m elevation

15.10.3 Mineral Resource Sensitivity

The grade tonnage distribution for Indicated and Inferred Resources at Black Fox are shown in Figures 15-3 through 15-6. The Resources have been subdivided based on the relative location above the mine elevation of 9,814.5m. This level represents the lowest bench potentially accessible by open pit mining; all Resources below this level would potentially be extracted by underground methods.

15.11 Reserve Estimation

The orientation, proximity to the surface, and geological controls of the Black Fox ore body will require mining of the ore reserves with open pit and underground mining techniques. Hence, the ore reserves are subdivided into open pit and underground categories.

15.11.1 Reserve Statement

Table 15.11.1.1: Open Pit and Underground Ore Reserve Statement

Classification	Category	Resource (kt)	Grade (gpt)	Gold (koz)
Open Pit	Proven	0	0	0
	Probable	4,350	5.2	730
	Proven and Probable	4,350	5.2	730
Underground	Proven	0	0	0
	Probable	2,110	8.8	600
	Proven and Probable	2,110	8.8	600
Combined	**Total Proven**	**0**	**0**	**0**
	Total Probable	**6,460**	**8.8**	**1330**
	Total Proven and Probable	**6,460**	**6.4**	**1,330**

These reserves are based on a gold price of $650/oz. A cut-off grade of 0.88gpt is used in the open pit and 3gpt in the underground design.

15.11.2 Conversion of Mineral Resources to Mineral Reserves

Open Pit

The conversion of mineral resources to open pit ore reserves requires accumulated knowledge achieved through pit optimization, pit design and associated modifying parameters.

Through the process of pit optimization and pit layout, a series of pit solid triangulations are created forming the basis for mine reserves. Figure 15-7 Illustrates the overall process flow and logic behind the formulation of mine reserves specific to the Black fox Project.

Two aspects are related for the conversion of resources to reserves:

- The ore extraction method(s) used in relation to the orebody characteristics which determine mining dilution and recovery; and
- Project operating costs and cut-off grade.

In accordance with the CIM classification system only Measured and Indicated resource categories can be converted to reserves (through inclusion within the open-pit mining limits). In all mineral reserve statements Inferred mineral resources are reported as waste.

Cut-off grade is a function of technical and economic parameters and defines the economic portion of the resource at the time of determination. Break even cut-off grade considers the total unit operating costs, including mining, processing and administration, process recovery, metal prices and additional costs for freight, smelting and/or refining. Where applicable, royalties are included in the calculation.

Once such a CoG is defined all the ore with a gold grade above this value should be considered as economically mineable. Ore feed to plant will have an average grade higher than the cut-off grade value, and this difference provides the profit (return on capital) for the business.

The typical expression for a break even (BE) gold CoG is (allowing for appropriate use of units):

$$\text{BE Cut-Off Grade} = \frac{\text{Total Unit Mining, Processing \& Administration Operating Costs}}{(\text{Au Price} - (\text{Royalty} + \text{Final Refining Costs})) \times \text{Process Recovery}}$$

The CoG used by Whittle to determine whether a block was ore or waste was reported as 0.88gpt Au. To keep consistency with what was used in the optimization, 0.88gpt was used to define ore and waste.

Underground

A cut and fill mining method is used as the basis for the underground mine design. Cut and fill is most suited to the orebody because:

- The "nuggety" and complex nature of the mineralization will require a high degree of geological control and mining flexibility; and
- The shallow 50º dip of the orebody precludes the use of longhole mining methods.

The underground stopes are designed by slicing the resource block model on 3m plan views. The design is constructed using only the indicated resource blocks above the cut-off grade. Lower grade indicated blocks are displayed during the design process but are only included in the design if higher-grade material could be added by including them. Polygons are digitized to define the stope outlines on the 3m plan views. The polygons are then extruded 3m vertically to define the 3D stope shape.

During the estimation process inferred blocks within the stope designs are treated as mined tonnage with zero grade. Indicated blocks within the stope design with grade below the cut-off are included in the gold estimation.

The reserve estimate includes 17% planned dilution in the form of footwall and hangingwall waste material that is mined within the designed stope shapes and inferred material in which the grade has been zeroed out.

A further 5% of zero grade unplanned dilution is added to the estimate to account for overbreak on the hangingwall and footwall sides of the stopes.

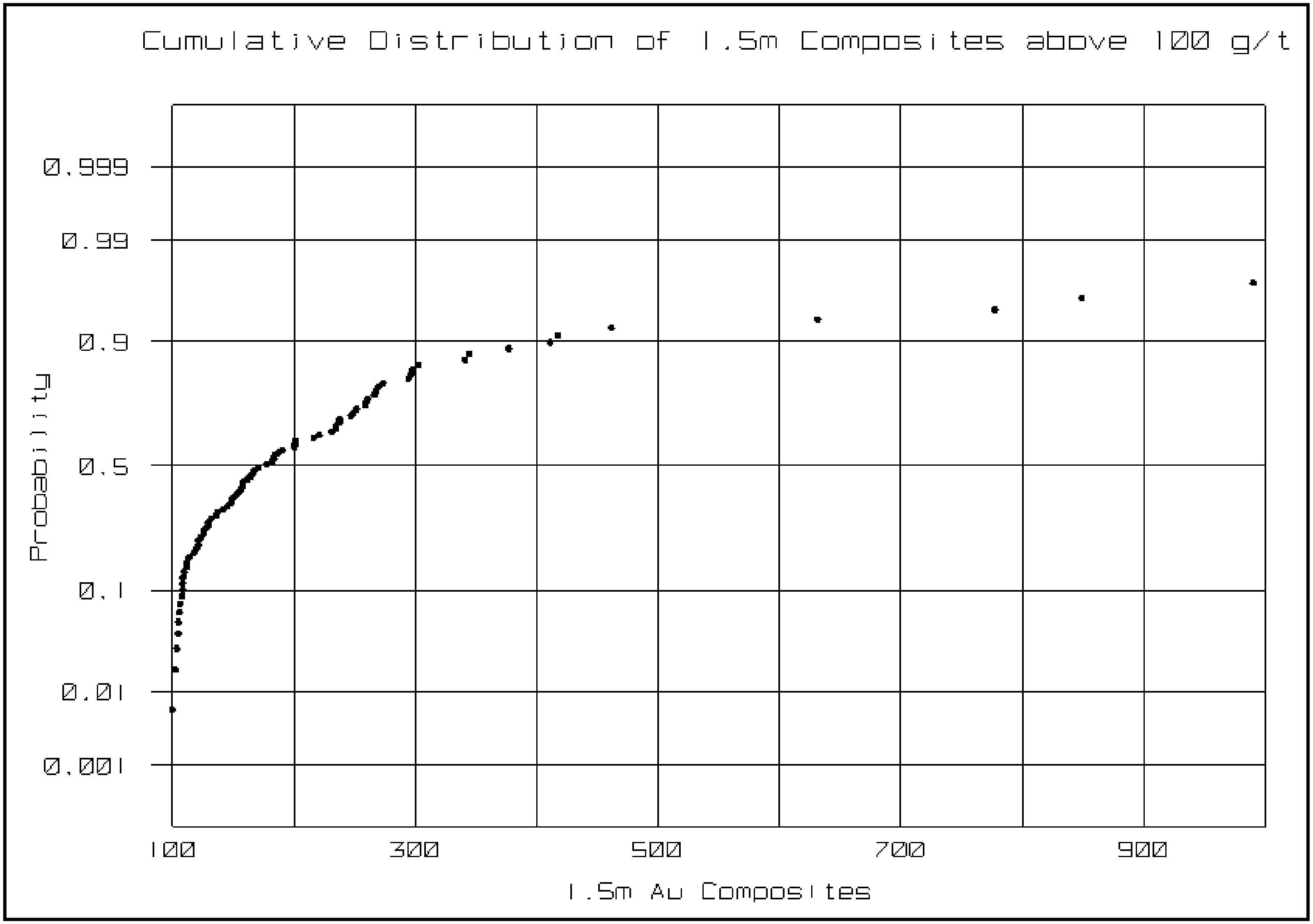
Table 15.11.2.1 presents the underground cut-off grade parameters and calculation. The mining milling and administration costs are based on the pre-Feasibility Study conclusions.

The mineral reserves are calculated based on a gold price of US$650/oz. A mill recovery of 95% is assumed and the refinery charge is US$2.50/oz. There are no royalties payable at the 100% owned property.

The calculated cut-off value of 3.3gpt was rounded to 3gpt for the mine reserve design process.

Table 15.11.2.1: Underground Cut-off Grade

Parameter	Unit	Amount
Mining cost	$/t	49.00
Milling cost	$/t	13.00
Admin cost	$/t	4.00
Total cost	**$/t**	**66.00**
Gold price	$/oz	650
Mill recovery		95%
Refinery Charges	$/oz	2.50
Net oz value	$/oz	647.5
Cut-off grade	gpt	3.30



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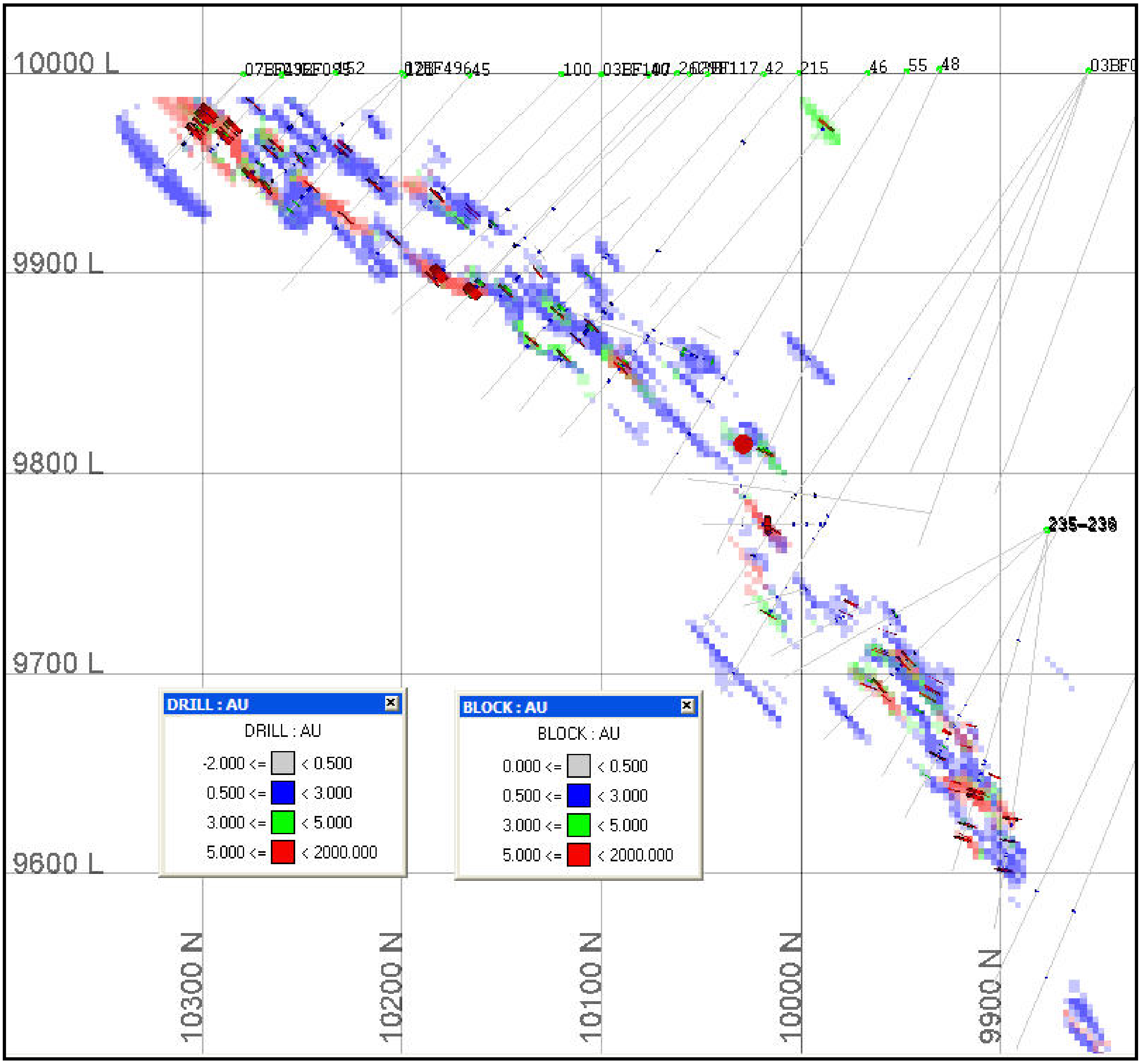
SRK Job No.: 144418

File Name: Figure 15-1.doc

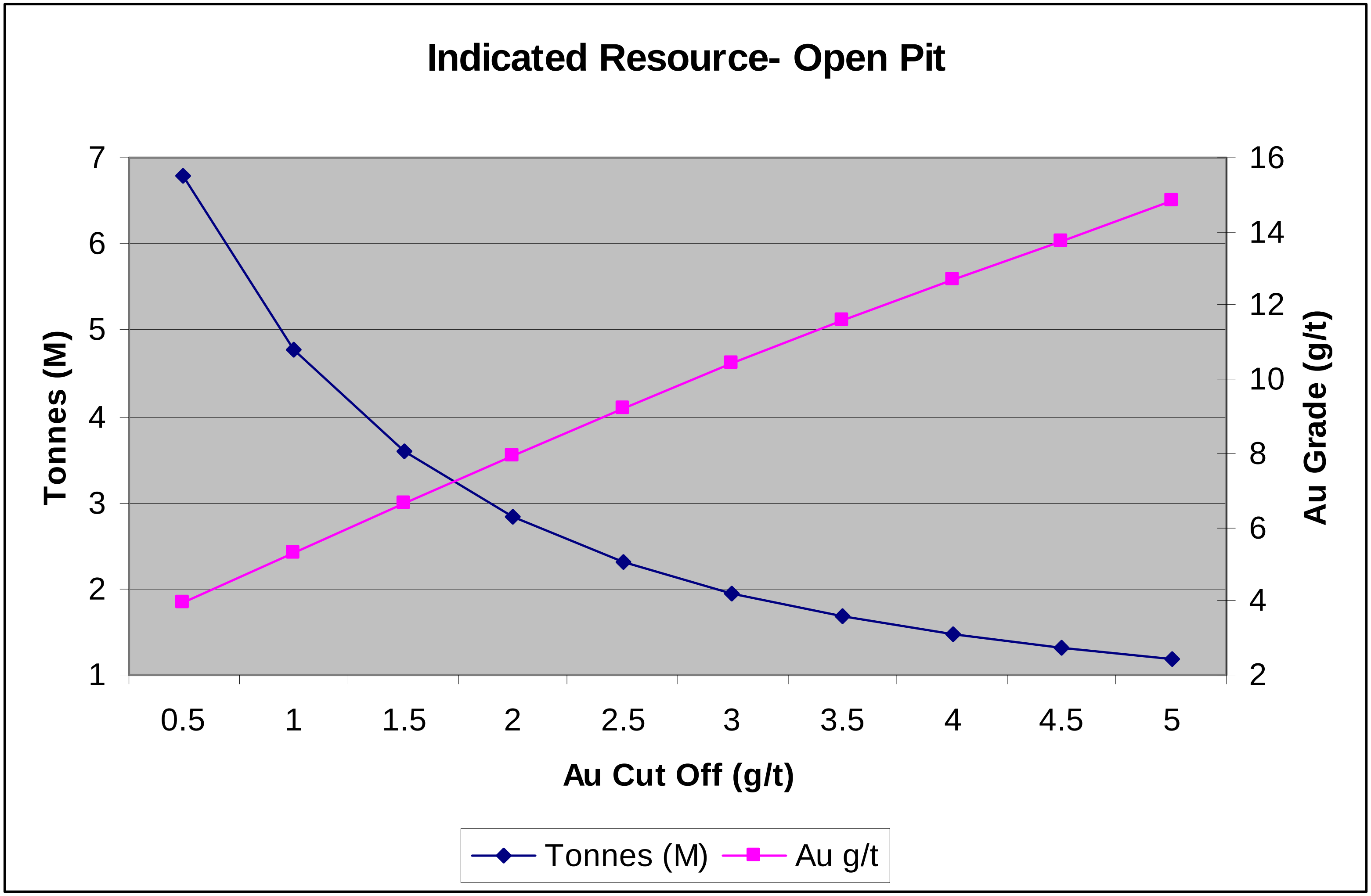
Black Fox,
Timmins, Ontario, Canada

Cumulative Frequency Plot of 1.5m Composites above 100g/t-Au in all Ore Types

Date: 02/13/08	Approved: BaS	Figure: 15-1



SRK Consulting Engineers and Scientists	**Black Fox,** **Timmins, Ontario, Canada**	**Black Fox Typical Block Model Cross-section 10300E Showing Distribution of Au in g/t**
SRK Job No.: 144418		Date: 02/18/08 — Approved: BaS — Figure: 15-2
File Name: Figure 15-2.doc		



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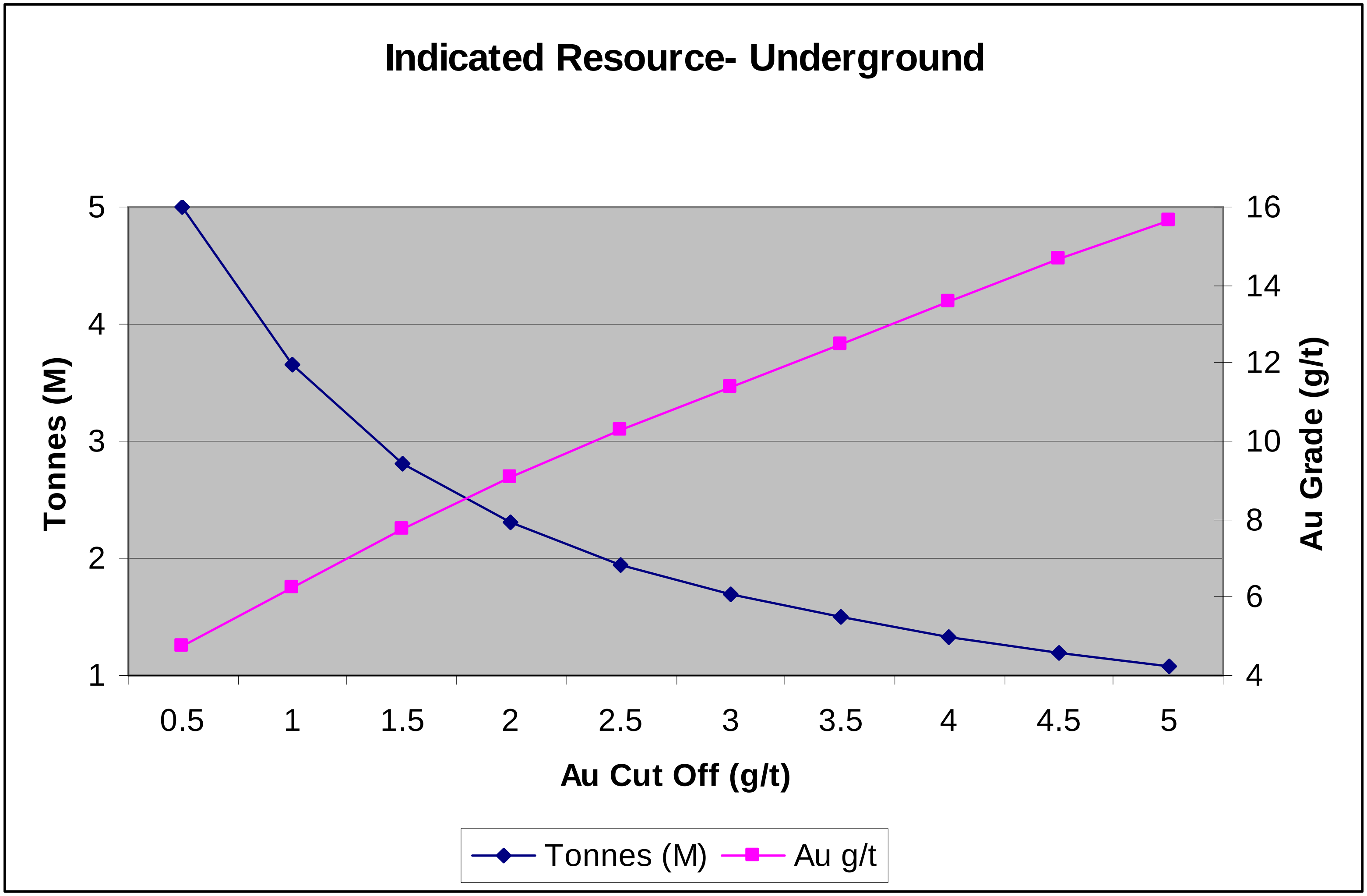
File Name: Figure 15-3.doc

Black Fox,
Timmins, Ontario, Canada

Grade Tonnage Curves for Indicated Open Pit Resources at Black Fox

Date: 02/13/08 | Approved: BaS | Figure: 15-3



SRK Consulting Engineers and Scientists	Black Fox, Timmins, Ontario, Canada	Grade Tonnage Curves for Indicated Underground Resources at Black Fox		
SRK Job No.: 144418		Date: 02/13/08	Approved: BaS	Figure: 15-4
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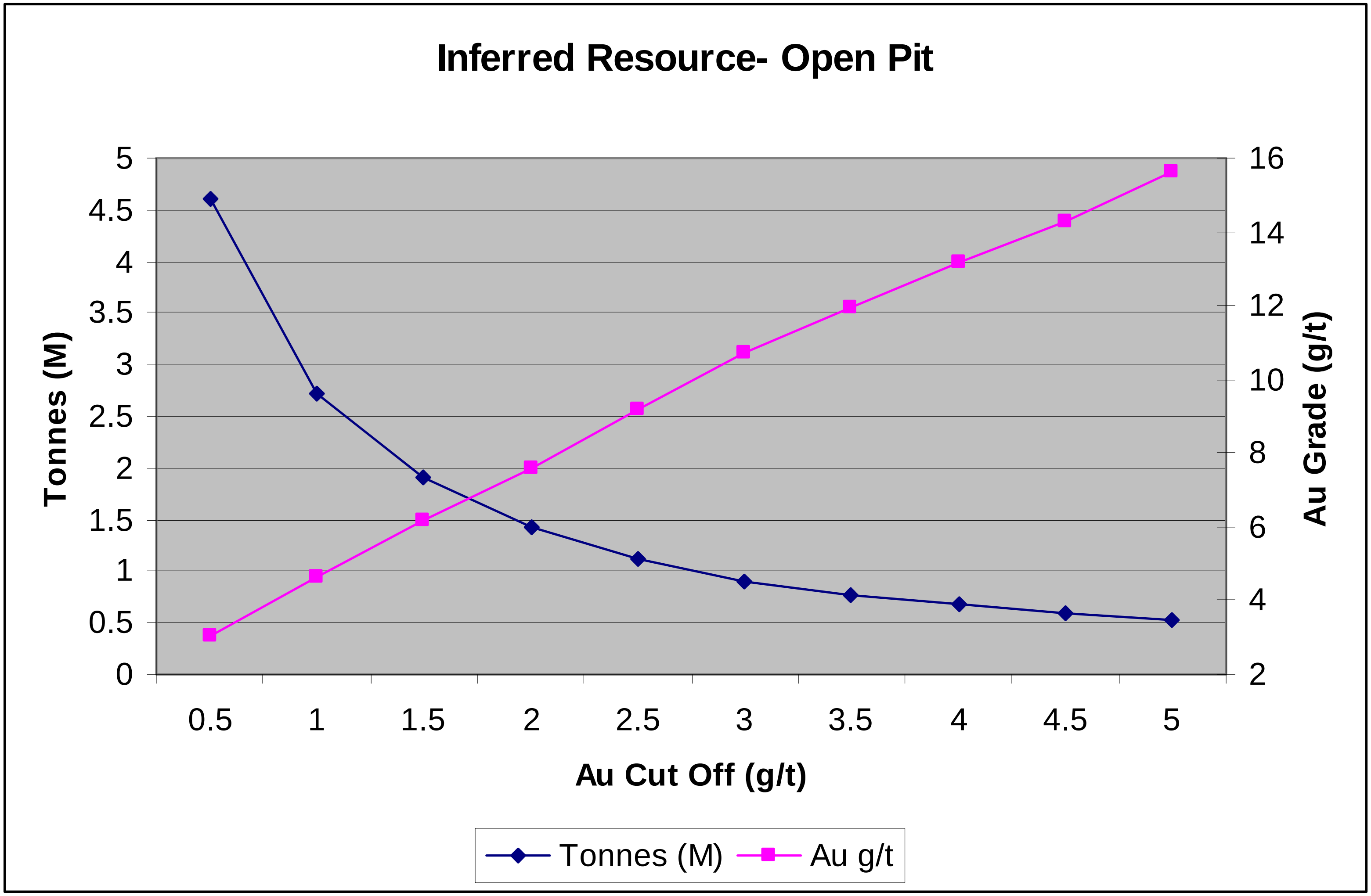

SRK Consulting Engineers and Scientists	Black Fox, Timmins, Ontario, Canada	**Grade Tonnage Curves for Inferred Open Pit Resources at Black Fox**
SRK Job No.: 144418		Date: 02/13/08 / Approved: BaS / Figure: 15-5
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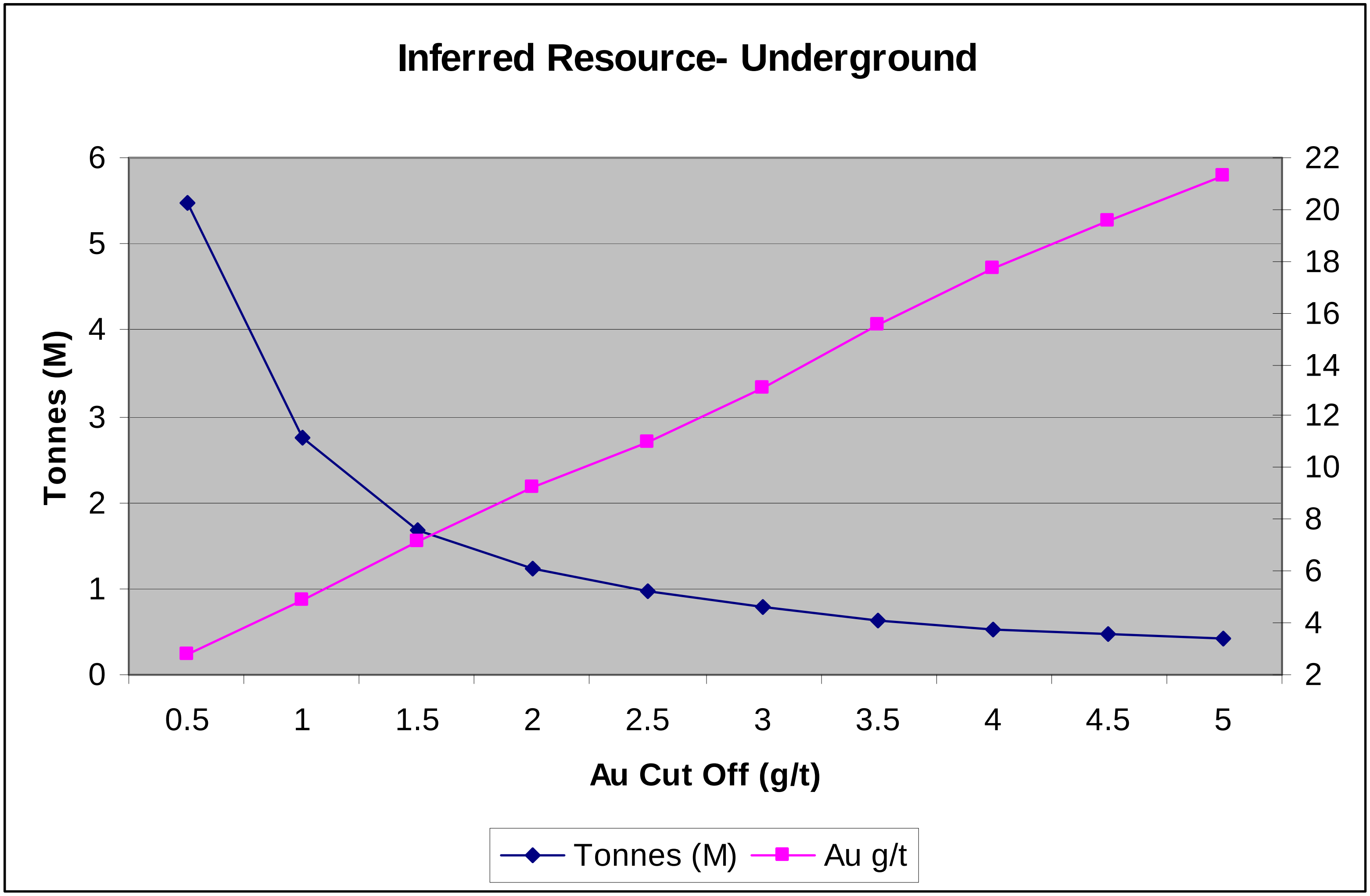

SRK Consulting Engineers and Scientists	Black Fox, Timmins, Ontario, Canada	Grade Tonnage Curves for Inferred Underground Resources at Black Fox
SRK Job No.: 144418		
File Name: Figure 15-6.doc		Date: 02/13/08 · Approved: BaS · Figure: 15-6

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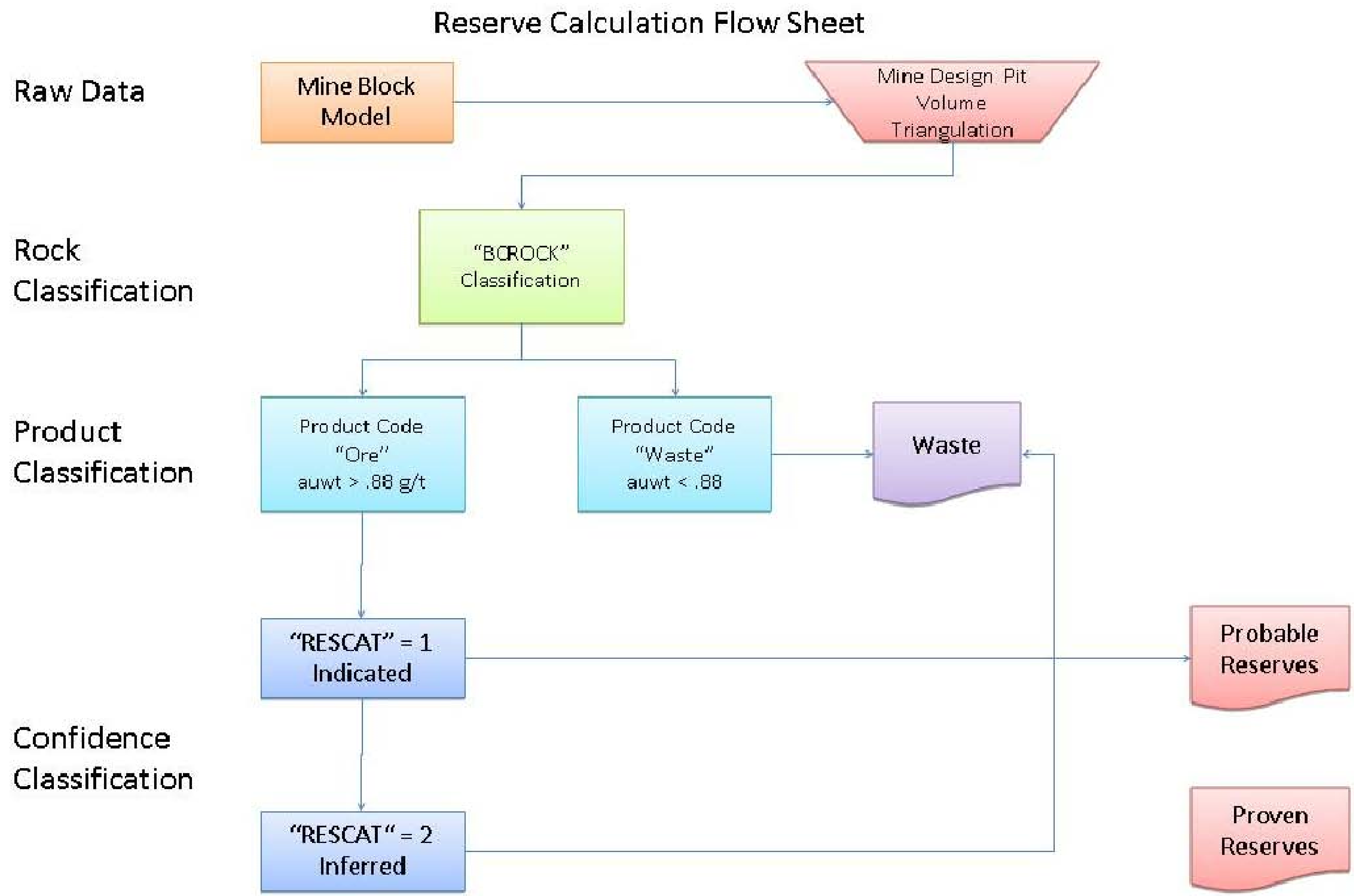

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BLACK FOX

RESERVE CALCULATION FLOW DIAGRAM

SRK JOB NO.: 0144418.00 / TASK 0005

FILE NAME: 0144418.00.Fig.15-7.Reserve.Calc.Flow.dwg

TIMMINS, ONTARIO, CANADA

DATE:	APPROVED:	FIGURE:	REVISION NO.
MAR. 2008	BS	15-7	A

16 Other Relevant Data and Information (Item 20)

The Black Fox orebody is a structurally defined zone of mineralization dipping at approximately 40º to 50º and is defined along strike. Mining will include open pit and underground mining operations. Ore will be milled using conventional crushing, grinding and CIP recovery technology.

The open pit operation designed for a 1,500tpd throughput will primarily use $4m^3$ and $10m^3$ hydraulic excavators loading 105t haul trucks. A $6.5m^3$ front-end loader will be used as backup to the excavators and will also be used for loading the backfill for the underground operations. Two drills will be purchased and a third one leased later in the project. The major support equipment will include two dozers, a grader and a water truck. The average stripping ratio based on the mine production schedule is 13:1 (waste to ore) with 4.4Mt or ore grading at 5.21gpt gold.

Black Fox underground mining will incorporate cut and fill mining, utilizing a mining cross-section of 3m high x 6m wide for the cut and a cemented rock fill (CRF) backfill material. Cut and fill was selected due the versatility of the method to allow the minimal amount of dilution while, meeting the production throughput target of 1,000tpd. Ore from underground will total 2.1Mt at an average grade of 8.8gpt.

The feasibility Study considered three milling options for the treatment of ores from Black Fox:

- Holt Mill – Toll milling up to 2,500tpd ore at SAS Holt Mill,
- Stock Mill – Milling up to 1,100tpd at the Stock Mill currently owned by SAS but the subject of a letter of intent signed in March 2008 between Apollo and SAS for Apollo to purchase the mill from SAS.
- Black Fox Mill – Design build and operate a 1,500tpd mill at the Black Fox mine.

Given these alternatives, it was determined that the best option for Apollo Gold, given the current understanding of the Black Fox deposit is to operate the Stock Mill at full capacity (nominal 1,100tpd) and toll mill the remaining mine production (nominal 1,400tpd) at the Holt Mill.

The Stock Mill, included the conventional unit processes of:

- Primary crushing;
- Closed circuit, single staged fine crushing;
- Two staged grinding;
- Pre-leach thickener and carbon columns;
- Leach and Carbon-in-Pulp (CIP) circuits;
- Carbon stripping and electrowinning; and
- Cyanide destruction.

The Black Fox Mill, while designed to feasibility-level, will be kept on reserve, until a time when additional reserves are discovered at Black Fox or when other, conditions make the construction of the Black Fox Mill economic.

16.1 Open Pit Mining

Mining operations have been separated into glacial overburden (Till) mining, which will be carried out by 3rd party contractors, and bedrock mining, which will be carried out by Apollo employees. The mining of bedrock has been separated into three distinct areas for calculating the equipment and costs required for each area. Mining in the waste, ore and old workings areas has been calculated using distinct parameters to account for the various methods and time that will be required for each area. Figure 16-1 shows the pit design and aerial photograph with phase areas for the Black Fox pit design.

Mining is expected to commence with Till removal in August 2008 with ore production beginning in January of 2009 at a rate of 1,000tpd through the 3rd quarter of 2010. After this time, permits should be in place and ore production can be ramped up to 1,500tpd for the remainder of the 9-year mine life. When full ore production is achieved in 2010, overburden stripping will be maximized at 35,000tpd for at least 1 year.

Ore will be delivered to a primary crusher located near the underground portal. Run of mine (RoM) ore stockpiles near the primary crusher are planned to provide continuity of ore delivery over short periods (with wheel loaders feeding the crusher if need be). Ore will be trucked to Apollo's Stock Mill and to St. Andrews' Holt Mill for processing.

Technical mine planning involved the creation of a mining block model, pit optimization at US$650/oz gold, detailed pit and dump design, production scheduling, haul profile calculation and generation of reserves. Information produced was then used for mine fleet sizing, costing, fleet performance, of which, results were rolled up into the SRK economic model.

The required mine equipment was calculated based on the open pit production schedule. The Open Pit equipment requirements were calculated taking into account four distinct areas:

- Glacial Till;
- Waste mining away from mineralization;
- Ore and waste mining near mineralization; and
- Open Pit mining in the old working areas.

16.1.1 Mining Methods

Glacial Till

The alluvial Till mining will be completed by third party contractors. Till material will be dumped in separate overburden dump locations to the north of highway 101E and to the west of Froome Lake.

Open Pit Waste Mining Away from Mineralization

In areas known to contain only waste, the drillhole spacing and drillhole depth will be different from areas containing mineralization. About 75% of the waste rock mined will be from areas containing only waste. Blasted waste rock will be mined by a $10m^3$ excavator and 105t haul trucks. A $6.5m^3$ front-end loader will be used as a backup for the excavator.

Open Pit Mining in the Mineralized Areas

The mineralized zones average 4.75m wide, but can be as narrow as 1.0m The minimization of dilution of the ore will be a critical element of the mining operation due to the characteristics of the orebody. For this reason, it is expected that the ore will be mined in 3m lifts or benches so that identification of ore blocks can be carried out with the most accuracy and the material mined with the minimum of dilution. It is expected that the drill cuttings from all blastholes will be sampled and assayed in order to provide the basis for ore grade control. The sampling requirements may dictate the spacing of the blastholes and this in turn would impact the blasthole diameter.

Mining in the mineralized area will be accomplished using a smaller $4.0m^3$ mass excavator to minimize dilution. For this study all of the ore was scheduled using the smaller excavator but in the larger ore zones the $10.0m^3$ excavator could be used to decrease the loading times. The 105t haul trucks will also be used for mining in the ore zones. The loading of the ore zones was restricted to daytime shifts only to provide better control in mining the ore.

Open Pit Mining through Existing Underground Working Areas

Additional factors have been added to the equipment requirement calculations to allow for the extra time and costs that will be encountered during the mining through the existing underground workings. The amount of material subject to costs was calculated as any block that was inside of or touching the existing workings. If a block was touching the existing workings, the full tonnage for that block was added to the total amount of material. The tonnage required for the blocks inside the old working was calculated at half the volume of the old workings backfilled with waste.

Mining in the existing underground workings was scheduled using the smaller $4.0m^3$ mass excavator. The production rate for mining in these areas was calculated at 65% of the normal production rate to allow for the extra time that is expected in these areas. The 105t haul trucks will also be used for mining in the ore zones. The loading of the old working areas was restricted to daytime shifts only for safety concerns.

16.1.2 Pit Slope Studies

The ore body is contained within the rocks of the Destor-Porcupine Fault Zone (DPFZ) which strikes generally east-west through the project area and dips between 40º to 50º to the south. Rock types expected to be encountered include ultramafic and mafic volcanic and sediments that have been altered and/or bleached and include such alteration assemblages of chlorite, talc-chlorite, ankerite, and sericite. Faulting also typically follows the dip of the DPFZ.

The Black Fox Open Pit is anticipated to be approximately 1km long along its long axis which is oriented approximately east-west, parallel to the strike of the ore body and over 500m wide and 150 m deep. The proposed north wall of the open pit is expected to follow the footwall of the DPFZ, while the south wall will straddle across hangingwall fault of the deformation zone. The geotechnical analysis of the recent investigations and review of previous reports indicates the rock mass possesses both competent and weak rock mass strengths, with the chloritized and foliated lithologic groups such as the chloritic altered ultramafics (CUV, TUV, AUV) specifically displaying anisotropic strengths due to foliation (observed during point load testing).

Structural domains appear not only be lithologically dependant, but also dependant upon the lithology location within the DPFZ, so separate geotechnical/structural domains appear to exist inside the DPFZ as well as outside the zone.

To complicate stability issues, there are current underground workings that are anticipated to intersect the intermediate and final pit walls as well as exist under the “floor” of the pit. The workings and the procedures used to deal with the stability issues during mine operation can have a significant impact on operational safety as well as the pit wall stability.

The author has first hand knowledge of these issues and has had to develop procedures to address personnel safety, equipment safety as well as overall stability of the pit walls. The operational procedures and stability hazards of working within underground openings must be addressed in order to fully appreciate the cost implications.

With the above issues in mind, it is recommended that the open pit shells be developed based on the design criteria in Table 16.1.2.1 below, and then the slopes optimized to take into consideration the lithologic sequence as well as underground workings that occur behind the pit walls. The analysis must consider the staged pit designs as well as final pit designs.

Table 16.1.2.1 below identifies both “Achievable” and “Optimistic” design parameters. Achievable designs are considered practical based on current industry experience, given current industry standards including some wall control blasting. Optimistic designs are aggressive and will require more expensive mining practices, wall control blast techniques, and careful operational procedures in order for these walls to be successful.

Table 16.1.2.1: Geotechnical Information

		Final Walls Achievable Configuration (18m high benches)		Stage I Pit Slope Optimistic Configuration*	
Domain	Wall Dip Direction	Bench Face Angle/ Berm Width	Inter Ramp Angle	Bench Face Angle/ Berm Width	Inter Ramp Angle
I	190º	55°/8 m	41°	60°/6-10m (alternating)	44°
II	220º	55°/8 m	41°	55°/8m	41°*
III	180º	60°/10 m	41°	60°/10m	41°*
IV	220º	55°/8 m	41°	55°/8m	41°*
V	000º	75°/8 m	54.5°	75°/8m	54.5°
VI	045^0	75°/8 m	54.5°	75°/8m	54.5°
VII	120º	65°/8 m	48º	70º/8m	51º

* Cannot be increased due to a possibility of day-lighting fault structures.

16.1.3 Pit Optimization

Pit optimization was carried out using Whittle 4.1™ optimization software using input parameters defined by SRK and feasibility contributors in January 2008.

Results were evaluated to determine the largest economic or Ultimate pit size, Optimum pit size and Optimum pit size above a mine elevation of RL 9,814.

There were several restrictions placed on optimization runs:

- Waste rock was to be minimized due to possible arsenic leaching from waste dumps and planned storage capacity;

- The definition of open pit and underground mining required space for a crown pillar;
- The requirement that one small pit be permitted and mined before mining to the optimization limits;
- Consideration of underground workings and Till removal costs; and
- Pit area restrictions from highway, lease boundaries and planned infrastructure.

16.1.4 Whittle™ Model Parameters

Table 16.1.4.1 illustrates the block dimensions and geotechnical zones of the exported Vulcan™ mine block model into Whittle™. Reblocking converted the geological block model size from 3m x 3m x 3m to 6m x 6m x 6m.

Table 16.1.4.1: Whittle™ Parameters

Whittle Parameter	Type	Value
Base Units		
	Au	g
	Mass	t
Block Model Dimensions		
	Origin (mine grid)	
	x	9700
	y	9500
	z	9400
	Geological	
	X	3
	Y	3
	Z	3
	No. X	317
	No. Y	284
	No. Z	209
	Reblocked	
	X	6
	Y	6
	Z	6
	No. X	159
	No. Y	142
	No. Z	105
Slope		
	Profile 1 (Zone 1)	41^0
	Profile 2 (Zone 2)	41^0
	Profile 3 (Zone 3)	41^0
	Profile 4 (Zone 4)	41^0
	Profile 5 (Zone 5)	54.5^0
	Profile 6 (Zone 6)	54.5^0
	Profile 7 (Zone 7)	48^0
	Profile 8 (Zone 10)	20^0
	Profile 9 (Zone 45)	45^0

16.1.5 Whittle™ Economic Parameters and Cost Adjustment Factors

With the complexity of mining in and around underground workings, a mining cost modification was made to blocks with the rock codes of "ptmn" (Block touches underground mining triangulation) and "ugmn" (Block completely encapsulated by underground mining triangulation). On average it was estimated a 37.8% increase in mining cost would result from additional probe drilling and loss of production associated with methodologies to improve safety around historical voids. For example: To mine a “ptmn” or “ugmn” block it would cost US$1.49/t x 1.378 = US$2.05/t. Table 16.1.5.1 shows the percentage variation of void mining cost associated with underground and partial mining.

Table 16.1.5.1: Void Mining Cost Adjustment Breakdown

Mining Variable	Units	Value
Total Material Mined	**(kt)**	**1,400**
General Mine Expense	% Increase US$/t Mined *	25
Drilling	% Increase US$/t Mined ***	119.6
Blasting	% Increase US$/t Mined ***	58.6
Loading	% Increase US$/t Mined **	25
Hauling	% Increase US$/t Mined **	25
Support	% Increase US$/t Mined **	25
Weighted Average	**%**	**37.8**

* Cost factor added for efficiency
** Detailed calculation

For optimization purposes, Till was included as an owner operator cost and was modeled on the base mining cost without a drill and blast penalty. The cost adjustment factor of 0.79 realized a Till mining cost of US$1.17/t.

A discount rate of zero was applied which removed any time cost of mining influence in the optimization and is handled in the SRK economic model.

Each economic rock type was used in a mineral processing path with a gold recovery of 95%. Till and Air did not contribute to optimization cash flow results.

Revenue factors used the default 0.02 step of gold price away from the base US$650/oz gold price. This represented an analysis of pits from US$195/oz to US$1,300/oz for gold.

Table 16.1.5.2: Whittle™ Parameters

Whittle™ Parameter	Type	Value
Mining Cost		
	Reference Mining Cost	US$1.49/t
	Mining Recovery Fraction	1
	Mining Dilution Factor	1
	Cost Adjustment Factors	
	TILL	0.79
	UGMN	1.37
	PTMN	1.37
Processing Cost		
	Process Name	Mill
	Rocktype 1	NGEO
	Rocktype 2	CUV
	Rocktype 3	PMV
	Rocktype 4	UV
	Rocktype 5	TUV
	Rocktype 6	MV
	Rocktype 7	AUV
	Rocktype 8	CGY
	Rocktype 9	MI
	Rocktype 10	BMV
	Rocktype 11	SUV
	Rocktype 12	FZ
	Rocktype 13	CGR
	Rocktype 14	QV
	Rocktype 15	FI
	Rocktype 16	BUV
	Rocktype 17	SEDS
	Rocktype 18	UGMN
	Rocktype 19	PTMN
	Ore Selection Method	Cut-off
	Processing Cost	18.88 $/ore t
	au recovery	0.95
Selling Cost		
	au units	oz
	au price	$650
Optimization		
	Revenue Factor Range	0.3-2, 86 factors
Operational Scenario - Time Costs		
	Initial Capital Cost	0
	Discount Rate Per Period	0
Operational Scenario - Limits		
	Processing Method Limits	360,000t

16.1.6 Optimal Pit Shell

Figure 16-2 shows the pit shells produced through optimization runs considering all blocks in the block model and those above RL 9,814. The pit shell selected as the base for pit design (Red triangulation in Figure 16-2) achieved the following results:

- Minimal risk to gold price fluctuations given conservative position on open pit value curve;
- Mining cost/ore-t is relatively low;

- Provision for crown pillar does not adversely effect optimization given ability to mine underground;
- Waste rock is kept to a minimum;
- No infringement on highway or lease boundaries;
- Minimal reliance on high-grade pods driving increased stripping; and
- No infringement on available dump space or infrastructure.

16.1.7 Pit Design

The Black Fox pit design utilized Vulcan™ software (v7.5) for crest, toe and ramp layout with dynamic application of berm and batter angles from block model variables. Using the Whittle™ optimal pit shell, the final pit design aimed to minimize stripping based on application of haul roads, staying within lease and infrastructure limits, and maintaining complex geotechnical constraints.

16.1.8 Pit Design Criteria

During the process of pit construction, several iterations were conducted and compared with pit designs achieved in the Black Fox pre-Feasibility Study. The culmination of this produced a final pit design that would:

- Keep the pit as tight as possible to the pit optimization shell on the NW wall;
- Provide at least 25m clearance from the highway without using retaining walls (Which would increase the slope above 20º in Till)
- Ensure the east wall did not extend into any disputed lease holding;
- Minimize ramps in the hanging wall and NW footwall;
- Place ramps on the NE footwall where the slope is relatively shallow, hence minimizing stripping;
- Place a 25m wide ramp to elevation RL 9,877 and then convert to a 15m wide one-way traffic ramp to the pit bottom;
- Incorporate pit wall berms and batter angles based on Table 16.1.2.1 and applied from block model zones;
- Facilitate an enlarged pit bottom on west side to extract additional waste and provide mining room for crown pillar construction; and
- Prevent “Doughnut” mining between mineralized areas

16.1.9 Bench Configuration

Geotechnical benches were based on an 18m inter-berm change in elevation. The final pit design was based on 6m projections with “zero berms” being applied every two projections and berm widths extracted from the block model on the third projection. This “triple benching” methodology did not restrict the pit design to geometry changes every 18m and maintained the overall geotechnical requirements for the deposit. Each 6m lift within the 18m geotechnical bench will be excavated using two 3m “Flitches”.

16.1.10 Haul Road Parameters

Haul roads were 25m wide for two way traffic and 15m for one-way traffic. A 25m wide ramp provides a truck width (6.39m) to running surface width ratio of about 3.5, which is considered safe. At elevation 9877, it was necessary to reduce road width to single lane traffic to minimize excessive waste stripping or loss of recoverable ore.

Roads had a maximum gradient of 10% assigned to the shortest distance along a ramp, which prevents gradient rules being broken around corners. The inside circumference of a ramp may be greater than 10% if the gradient is applied to the ramp centerline or high wall.

16.1.11 Phase Design

It was necessary to break the final pit design into 3 phases. Phase 1 was defined by a polygon area (In which the phase 1 pit was to stay within) for permitting reasons but would still merge with common walls of the final pit. The phase 2 pit provided a bridge for scheduling purposes between phase 1 and the final pit with phase intersections dictated by final pit design. Figure 16-1 shows a plan view with aerial photo of where each phase intersection on topography resides.

Phase 1 pit bottom terminated at elevation RL 9,907 and Phase 2 terminated at elevation RL 9,943 while phase 3 terminated at the final pit floor. Table 16.1.11.1 details the ore and waste tonnages defined by the different phases.

Table 16.1.11.1: Phase Tonnage

Phase	Au (gpt)	Ore Mass (t)	Waste Mass (t)
Phase 1	6.83	964,442	7,669,253
Phase 2	4.25	349,944	4,832,348
Phase 3	4.79	3,030,091	33,557,894

Till material was to be mined using contractors at a rate of 15,000tpd for phase 1 and required 20,000tpd for phase 2 and 3. Although no haul profiles or waste schedules were performed for Till, it was necessary to schedule its removal for mine precedence reasons.

16.1.12 Bedrock Dump Design

The Bedrock dump design target volume was limited to Bedrock material produced in the production schedule that would be mined by Apollo. Till material which is expected to be mined using third party contractors was not included in “Bedrock” dump design.

The dump design for Black Fox was broken into two parts. Waste from the first two years of mining was required to stay within a specified polygon used for permitting. Access to the tip head for this dump was via the crusher and extended horizontally from the current outcrop elevation RL 10,020. The initial dump design ultimately lies within the total dump footprint so no benches were included in the design. Table 16.1.12.1 illustrates the parameters assigned to the final dump design configuration.

The final dump design used the full area set aside for waste dump disposal in the pre feasibility Study and was modified to include dual access to the tip head locations. Access to the tip head via the crusher established during phase 1 mining would necessitate a long haul, so the final dump design had the tip head access as close as possible to the termination point of the ramps for Phase 2 and 3. This allowed two entry points to the dump throughout the mine life and minimized haul distances. The final dump design reached elevation RL 10,065 and contained room for 23.1Mm^3 of fill.

Table 16.1.12.1: Dump Design Parameters

Parameter	Units	Value
Berm Width	m	7m
Lift Height	m	10m
Batter Angle	º	37
Overall Slope	Ratio	1:2
Ramp Width	m	25
Ramp Grade	%	10

The final dump volume is more than adequate room to accept all pit design waste and has plenty of room for vertical expansion.

The final dump toe generally follows an outcrop boundary between hard rock and loose Till material. Since geotechnical holes are sparse on the SW dump face, careful monitoring is needed if the material dumped on this Till material leads to geotechnical instability.

16.1.13 Pit Production Schedule

Production scheduling was carried out using Vulcan™ (v7.5) and its scheduling package Chronos™. The schedule was constructed around a 1,000tpd mill feed for the first 9 quarters increasing to 1,500tpd for the remainder of the mine life. The amount of waste stripping was maximized at 35,000tpd. Ore was defined using a 0.9gpt cut-off as indicated from pit optimization work and scheduling units were broken into Till and Bedrock blocks within each phase. The production schedule was used to estimate the quantities of waste material produced each year for dump design and as an estimation for annual haul cycle times and distances.

The production schedule was targeted quarterly with period 1 being slightly shorter due to the schedule commencement in August 2008. Open pit mining terminated in the third quarter of 2017 giving a mine life of 9 years.

Figure 16-3 shows the production schedule results with grade, ore and waste tonnes per day.

16.1.14 Grade Schedule

Grade and tonnage information related to mining blocks used in the production schedule are stored in a “reserves” spreadsheet within Chronos™. Each key field or solid triangulation contains a reserve classification of either ore or waste based on a cut-off grade (CoG) of 0.9gpt Au. This Product Code is then classified according to the proportional volume of geological rock type within each product. The result of this dual “folding” of reserves is the reporting of tonnes and grade for each product and rock type. This disseminated grade and tonnage information is combined (weighted average of relevant variables) into fields used for scheduling targets.

Table 16.1.14.1: Annual Production Summaries

Production Year	Total Ore (t)	Total Au (gpt)	Total InSitu (oz)	Bedrock Waste (t)	Till Waste (t)	Total Waste (t)
2008	763	13.48	331	15,147	2,265,000	2,280,147
2009	364,295	9.09	106,434	3,312,189	1,304,365	4,616,553
2010	407,761	5.51	72,226	5,068,258	5,440,000	10,508,258
2011	541,515	4.90	85,346	11,595,463	1,797,884	13,393,347
2012	543,000	3.65	63,679	7,693,586	-	7,693,586
2013	541,500	4.39	76,389	5,828,425	-	5,828,425
2014	541,500	4.74	82,547	4,371,930	-	4,371,930
2015	541,500	5.42	94,319	3,285,412	-	3,285,412
2016	543,000	5.51	96,240	3,281,526	-	3,281,526
2017	324,803	4.89	51,036	1,607,559	-	1,607,559
Total	**4,349,636**	**5.21**	**728,547**	**46,059,494**	**10,807,248**	**56,866,743**

16.1.15 Waste Schedule

To calculate the required annual dump volume, a loose density of $2.0t/m^3$ was applied to the generated waste tonnage produced minus material destined for tailings construction.

For each production year, an incremental solid triangulation was created within the total waste dump design. The centroid of this dump shape was used as a terminating point for the digitized waste haulage profile.

Table 16.1.15.1: Annual Waste Schedule

Production Year	Phase 1	Phase 2	Phase 3	Target Waste Tonnage (t)	Phase 1	Phase 2	Phase 3	Target Waste Volume (m^3)	Tailings Construction	Cumulative
2008	15,147	-	-	15,147	7,574	-	-	7,574		7,574
2009	3,299,295	-	12,894	3,312,189	1,649,647	-	6,447	1,656,094	548,000	1,115,668
2010	1,915,618	2,791,832	360,808	5,068,258	957,809	1,395,916	180,404	2,534,129		3,649,797
2011	1,823,321	2,040,516	7,731,627	11,595,463	911,661	1,020,258	3,865,813	5,797,732		9,447,529
2012	615,872	-	7,077,714	7,693,586	307,936	-	3,538,857	3,846,793		13,294,322
2013	-	-	5,828,425	5,828,425	-	-	2,914,212	2,914,212	672,000	15,536,534
2014	-	-	4,371,930	4,371,930	-	-	2,185,965	2,185,965		17,722,499
2015	-	-	3,285,412	3,285,412	-	-	1,642,706	1,642,706		19,365,205
2016	-	-	3,281,526	3,281,526	-	-	1,640,763	1,640,763		21,005,968
2017	-	-	1,607,559	1,607,559	-	-	803,779	803,779		21,809,747
Grand Total	**7,669,253**	**4,832,348**	**33,557,894**	**46,059,494**	**3,834,627**	**2,416,174**	**16,778,947**	**23,029,747**	**1,220,000**	

16.1.16 Haulage Schedule

Haulage calculations based on the production schedule were estimated using Vulcan™ (v7.5) haul profile software and were used to calculate annual distance and cycle times for ore and waste. Haul profiles were calculated by digitizing ore and waste profiles from the pit haul road to either a dump centroid or crusher location. Blocks on a given bench elevation, in each phase, had their distances estimated to a “pit road” and added to the digitized haul routes at the pit exit. The result of this process was each block flagged as ore and waste received haul distance and cycle time values. These values then acted as a value variables and were subsequently reported according to the production schedule.

Table 16.1.16.1: Base Distance and Cycle Times without Delays or Efficiencies

Production Year	1-Way Waste Distance (m)	1-Way Ore Distance (m)	Ore Cycle Time (min)	Waste Cycle Time (min)
2009	1,552	1,570	4.46	5.83
2010	1,439	1,863	6.10	5.82
2011	1,436	2,385	7.65	6.50
2012	1,880	2,666	8.27	8.45
2013	2,342	2,788	9.10	10.81
2014	2,407	2,837	9.60	11.36
2015	2,648	3,076	11.25	13.01
2016	2,849	3,264	12.76	14.44
2017	3,174	3,515	14.48	16.51
Grand Total	**1,989**	**2,698**	**9.33**	**9.18**

16.1.17 Phase Design Plots

Phase plots have been generated for annual pit and dump advance. Each annual phase plot details the expected site layout, pit floor elevation, dump elevation and haul routes for ore and waste during the year. Pit and dump progression was based on the production schedule. Figure 16-4 shows the final phase design plot.

16.1.18 Open Pit Productivity

Drilling & Blasting

The design assumptions used to determine drilling and blasting productivity are shown in Table 16.1.18.1.

Table 16.1.18.1: Drilling & Blasting Assumptions

Drilling and Blasting Parameters	Units	Void Areas	Ore Bedrock	Ore Bedrock	Waste Bedrock	Waste Bedrock	Wall Control Pattern		
							Buffer	Buffer	Preshear
Tonnage Factor	dmt/m^3	2.840	2.840	2.840	2.840	2.840	2.840	2.840	2.840
Blast Pattern Details									
Bench Height	m	6.00	3.00	3.00	3.00	6.00	6.00	6.00	12.00
Sub Drill	m	1.25	0.63	0.63	0.63	1.00	1.00	1.00	0.00
Diameter of Hole	mm	110.00	110.00	165.00	110.00	165.00	165.00	165.00	110.00
Staggered Pattern Spacing	m	3.25	3.25	4.25	3.25	5.00	5.25	5.25	1.40
Staggered Pattern Burden	m	3.25	3.25	4.25	3.25	5.00	5.25	5.25	1.40
Drill Equivalent Square Pattern	m	3.25	3.25	4.25	3.25	5.00	5.25	5.25	1.40
Hole Depth	m	7.25	3.63	3.63	3.63	7.00	7.00	7.00	12.00
Height of Stemming or Unloaded Length	m	2.70	1.35	1.75	1.70	3.50	3.75	3.75	
Material Quantity									
Volume Blasted/Hole	m^3	63	32	54	32	150	165	165	24
Tonnes Blasted/Hole	t	180	90	154	90	426	470	470	67
Powder Factor									
Density of Powder	g/cc	0.94	0.94	0.94	0.94	0.94	0.94	0.94	1.23
Loading Density	kg/m	8.95	8.95	20.14	8.95	20.14	20.14	20.14	11.69
Powder/hole	kg	40.73	20.37	37.77	17.23	70.50	65.46	65.46	6.01
Powder Factor	kg/t	0.226	0.226	0.245	0.191	0.165	0.139	0.139	0.090
Powder Factor	kg/bm^3	**0.643**	**0.643**	**0.697**	**0.544**	**0.470**	**0.396**	**0.396**	**0.256**
Drill Productivities									
Penetration Rate	m/hr	30.00	40.00	30.00	40.00	30.00	40.00	40.00	30.00
Penetration Rate	m/min	**0.50**	**0.67**	**0.50**	**0.67**	**0.50**	**0.67**	**0.67**	**0.50**
Cycle Time Estimate									
Drilling Time	min	14.50	5.44	7.25	5.44	14.00	10.50	10.50	24.00
Steel Handling Time	min	0.20	0.20	0.20	0.20	0.20	0.20	0.20	1.00
Set up Time	min	0.80	0.80	0.80	0.80	0.80	0.80	0.80	1.00
Total	**min**	**15.50**	**6.44**	**8.25**	**6.44**	**15.00**	**11.50**	**11.50**	**26.00**

Table 16.1.18.1: Open Pit Drill and Blasting Assumptions (Continued)

Drill Productivities (continued)	Schedule Data	Material Type	Void Areas	Ore Bedrock	Ore Bedrock	Waste Bedrock	Waste Bedrock	Wall Control Pattern Buffer	Wall Control Pattern Preshear
	Calendar Days	days/year	365	365	365	365	365	365	365
	Scheduled Shutdown	days/year							
	Unscheduled Days Down	days/year	15	15	15	15	15	15	15
	Mine Work Days	days/year	350	350	350	350	350	350	350
	Work Days / Week	days/year	7	7	7	7	7	7	7
	Shifts / Day	shifts/day	2	2	2	2	2	2	2
	Shifts / Week	shifts/week	14	14	14	14	14	14	14
	Scheduled Weeks / Year	weeks/year	50	50	50	50	50	50	50
	Shifts / Year	shifts/year	700	700	700	700	700	700	700
	Scheduled Hours / Shift	hours/shift	10.5	10.5	10.5	10.5	10.5	10.5	10.5
	Scheduled Hours / Year	hours/year	7,350	7,350	7,350	7,350	7,350	7,350	7,350
(T)	Total Theoretical	hours/year	7,665	7,665	7,665	7,665	7,665	7,665	7,665
(SU)	Scheduled & Unscheduled Shutdown	hours/year	315	315	315	315	315	315	315
Standby									
	Lunch Break	hours/shift	0.50	0.50	0.50	0.50	0.50	0.50	0.50
	Shift Start / Shutdown	hours/shift	0.40	0.40	0.40	0.40	0.40	0.40	0.40
	Coffee Breaks	hours/shift	0.50	0.50	0.50	0.50	0.50	0.50	0.50
	Miscellaneous - Blasting & Moves	hours/shift	0.20	0.20	0.20	0.20	0.20	0.20	0.20
	Total Standby	hours/shift	1.6	1.6	1.6	1.6	1.6	1.6	1.6
(S)	Total Standby	hours/year	1,120	1,120	1,120	1,120	1,120	1,120	1,120
	Available Working Hours	hours/day	20.8	20.8	20.8	20.8	20.8	20.8	20.8
	Available Working Hours	hours/year	6,230	6,230	6,230	6,230	6,230	6,230	6,230
Annual Hours									
(T)	Total Theoretical	hours/year	7,665	7,665	7,665	7,665	7,665	7,665	7,665
(S)	Total Standby	hours/year	1,120	1,120	1,120	1,120	1,120	1,120	1,120
(SU)	Scheduled & Unscheduled Shutdown	hours/year	315	315	315	315	315	315	315
(W)+(R)	Work + Repair = (T-S-SU)	hours/year	6,230	6,230	6,230	6,230	6,230	6,230	6,230
(W)	Work = MA x (T-S-SU)	hours/year	4,984	4,984	4,984	4,984	4,984	4,984	4,984
Mechanical Availability Definition									
	Scheduled Downtime	shifts/year	35	35	35	35	35	35	35
	Scheduled Downtime	hours/year	367.5	367.5	367.5	367.5	367.5	367.5	367.5
	Scheduled Downtime	%	5.0%	5.0%	5.0%	5.0%	5.0%	5.0%	5.0%
	Unscheduled Downtime	%	15.0%	15.0%	15.0%	15.0%	15.0%	15.0%	15.0%
	Total Downtime	%	20.0%	20.0%	20.0%	20.0%	20.0%	20.0%	20.0%
	Shifts Available for Scheduling	shifts	665	665	665	665	665	665	665
(MA)	Mechanical Availability	%	80.0%	80.0%	80.0%	80.0%	80.0%	80.0%	80.0%
Physical Availability									
(PA)	Physical Availability = (W+S)/T		79.6%	79.6%	79.6%	79.6%	79.6%	79.6%	79.6%
Use of Availability									
(UA)	Use of Availability = W/(W+S)		81.7%	81.7%	81.7%	81.7%	81.7%	81.7%	81.7%
Effective Utilization									
(EU)	Effective Utilization = PA x UA		65.0%	65.0%	65.0%	65.0%	65.0%	65.0%	65.0%
Annual Production									
	Work Hours / Year	hours/year	4,984	4,984	4,984	4,984	4,984	4,984	4,984
	Operating Efficiency	%	80.0%	80.0%	80.0%	80.0%	80.0%	80.0%	80.0%
	Production Hours / Year	hours/year	3,987	3,987	3,987	3,987	3,987	3,987	3,987
	Scheduled Shifts / Year (from below)	shifts/year	665	665	665	665	665	665	665
	Work Shifts/Year	shifts/year	475	475	475	475	475	475	475
	Work Hours / Shift	hrs	8.9	8.9	8.9	8.9	8.9	8.9	8.9
	Drilling Hours per Shift	drill hrs/shift	7.1	7.1	7.1	7.1	7.1	7.1	7.1
	Drilling Minutes per Shift	drill min/shift	427.2	427.2	427.2	427.2	427.2	427.2	427.2
	Drilling Minutes per hole	drill min/hole	15.5	6.4	8.3	6.4	15.0	11.5	26.0
	Holes drilled per Shift	holes/shift	27.6	66.4	51.8	66.4	28.5	37.1	16.4
	Metres Drilled per Shift	m/shift	199.82	240.56	187.71	240.56	199.36	260.03	197.17
	Tonnes Drilled per Shift	t/shift	4,961	5,972	7,969	5,972	12,132	17,447	1,098
	Tonnes Drilled per Year	(kt)	2,355	2,835	3,783	2,835	5,759	8,282	521
	Production / Scheduled Work Hours	tph	472.4	568.8	758.9	568.8	1,155.5	1,661.6	104.5
	Production / Scheduled Prod. Hours	tph	590.5	711.0	948.7	711.0	1,444.3	2,077.0	130.7

Open Pit Loading Productivity

The design assumptions used to determine loading productivity, including the underground backfill loading and feeding the crusher, are shown in Table 16.1.18.2.

Table 16.1.18.2: Open Pit Loading Productivity

Loading Parameters & Truck Match Calculation Schedule Data		Material Type	$4.0m^3$ Excavator Waste 105t	Ore 105t	Rock 105t	$6.5m^3$ Loader Waste 105t	Ore 105t	Other $6.5m^3$
	Calendar Days	days/year	365	365	365	365	365	365
	Unscheduled Days Down - weather	days/year	15	15	15	15	15	15
	Mine Work Days	days/year	350	350	350	350	350	350
	Work Days / Week	days/year	7	7	7	7	7	7
	Shifts / Day	shifts/day	1	1	2	2	2	2
	Shifts / Week	shifts/week	7	7	14	14	14	14
	Scheduled Weeks / Year	weeks/year	50	50	50	50	50	50
	Shifts / Year	shifts/year	350	350	700	700	700	700
	Scheduled Hours / Shift	hours/shift	10.5	10.5	10.5	10.5	10.5	10.5
	Scheduled Hours / Year	hours/year	3,675	3,675	7,350	7,350	7,350	7,350
(T)	**Total Theoretical**	**hours/year**	**3,833**	**3,833**	**7,665**	**7,665**	**7,665**	**7,665**
(SU)	Scheduled & Unscheduled Shutdown	hours/year	158	158	315	315	315	315
Standby	Lunch Break	hours/shift	0.50	0.50	0.50	0.50	0.50	0.50
	Shift Start / Shutdown	hours/shift	0.40	0.40	0.40	0.40	0.40	0.40
	Coffee Breaks	hours/shift	0.50	0.50	0.50	0.50	0.50	0.50
	Miscellaneous - Blasting & Moves	hours/shift	0.25	0.25	0.25	0.25	0.25	
	Total Standby	**hours/shift**	**1.7**	**1.7**	**1.7**	**1.7**	**1.7**	**1.4**
(S)	**Total Standby**	**hours/year**	**578**	**578**	**1,155**	**1,155**	**1,155**	**980**
	Available Working Hours	hours/day	22.4	22.4	20.7	20.7	20.7	21.2
	Available Working Hours	hours/year	3,098	3,098	6,195	6,195	6,195	6,370
Annual Hours								
(T)	**Total Theoretical**	**hours/year**	**3,833**	**3,833**	**7,665**	**7,665**	**7,665**	**7,665**
(SU)	Scheduled & Unscheduled Shutdown	hours/year	158	158	315	315	315	315
(S)	**Total Standby**	**hours/year**	**578**	**578**	**1,155**	**1,155**	**1,155**	**980**
(W)+(R)	Work + Repair = (T-S-SU)	hours/year	3,098	3,098	6,195	6,195	6,195	6,370
(W)	Work = MA x (T-S-SU)	hours/year	2,633	2,633	5,266	5,266	5,266	5,415
Mechanical Availability								
	Scheduled Downtime	shifts/year	17.5	17.5	35	35	35	35
	Scheduled Downtime	hours/year	140	140	280	280	280	280
	Scheduled Downtime		5.0%	5.0%	5.0%	5.0%	5.0%	5.0%
	Unscheduled Downtime		10.0%	10.0%	10.0%	10.0%	10.0%	10.0%
	Total Downtime		**15.0%**	**15.0%**	**15.0%**	**15.0%**	**15.0%**	**15.0%**
	Shifts Available for Scheduling	shifts	333	333	665	665	665	665
(MA)	Mechanical Availability		85.0%	85.0%	85.0%	85.0%	85.0%	85.0%
(PA)	Physical Availability = (W+S)/T		83.8%	83.8%	83.8%	83.8%	83.8%	83.4%
(UA)	Use of Availability = W/(W+S)		82.0%	82.0%	82.0%	82.0%	82.0%	84.7%
(EU)	Effective Utilization = PA x UA		68.7%	68.7%	68.7%	68.7%	68.7%	70.6%
Annual Production								
(WH)	Work Hours / Year	hours/year	2,633	2,633	5,266	5,266	5,266	5,415
	Operating Efficiency - operation based		80.0%	80.0%	90.0%	85.0%	85.0%	80.0%
(PH)	Production Hours / Year	hours/year	2,106	2,106	4,739	4,476	4,476	4,332
(BC)	Bucket Capacity (heaped)	cm	4.00	4.00	10.00	6.50	6.50	6.50
(MW)	Material Weight	kg/bcm dry	2840	2840	2840	2840	2840	2,840
MWW	Material Weight Wet	kg/bcm wet	2850	2850	2850	2850	2850	2,860
(BF)	Bulk Factor (Swell Factor)		1.35	1.35	1.35	1.35	1.35	1.35
(MW1)	Material Weight = MW / BF	kg/lcm dry	2,103.7	2,103.7	2,103.7	2,103.7	2,103.7	2,103.7
(M)	Moisture		5.00%	5.00%	5.00%	5.00%	5.00%	5.00%
(FF)	Fill Factor		0.85	0.85	0.85	0.85	0.85	0.85
(EBC)	Effective Bucket Capacity = FF x BC	cm	3.40	3.40	8.50	5.53	5.53	5.53
(MW2)	Material Weight = MW1 / (1-M)	wmt/lcm	2.11	2.11	2.11	2.11	2.11	2.21
	Material Weight = MW2 x (1-M)	dmt/lcm	2.10	2.10	2.10	2.10	2.10	2.10
(TP)	Tonnes/Pass	wmt	7.18	7.18	17.94	11.66	11.66	12.23
(TC1)	Truck Size Capacity	m^3 heaped	78.0	78.0	60.1	60.1	60.1	42.0
(TC2)	Truck Size Capacity	wmt	105.0	105.0	105.0	105.0	105.0	70.0
(TPV)	Theoretical Passes = TC1/ EBC	passes	22.94	22.94	7.07	10.88	10.88	7.60
(TPT)	Theoretical Passes = TC2 / TP	passes	14.63	14.63	5.85	9.00	9.00	5.72
(AP)	Actual Passes = ROUND TPT	passes	14.0	14.0	6.0	9.0	9.0	6.0
(TL)	Truck Load - Volume = AP x EBC	cm	47.6	47.6	51.0	49.7	49.7	33.2
(TLS)	Truck Load for Simulation = AP x TP	wmt	100.5	100.5	107.7	105.0	105.0	73.4
(TLP)	Truck Load for Productivity	dmt	100.1	100.1	107.3	104.6	104.6	69.7
(TCU)	Truck Capacity Utilized = TLS / TC2	by weight	95.7%	95.7%	102.5%	100.0%	100.0%	104.9%
	Truck Capacity Utilized = TL / TC1	by volume	61.0%	61.0%	84.9%	82.7%	82.7%	78.9%
(AC)	Average Cycle Time	sec	27	27	27	39	39	180
(ST)	Truck Spot Time	sec	42	42	42	42	42	15
(LT)	Load Time per Truck = AP x AC + ST	sec	420	420	204	393	393	1095
(LT)	Load Time per Truck = AP x AC + ST	minutes	7.00	7.00	3.40	6.55	6.55	18.25

Table 16.1.18.2: Open Pit Loading Productivity (cont.)

Loading Parameters & Truck Match Calculation Schedule Data		Material Type	4.0m^3 Excavator Waste 105t	Ore 105t	Rock 105t	6.5m^3 Loader Waste 105t	Ore 105t	Other 6.5m^3
(MP)	Maximum Productivity = 60 / LT	trucks/hr	8.6	8.6	17.6	9.2	9.2	3.3
	Conversion = MP x TLP/ MW	bcm/hr	302.2	302.2	666.7	337.4	337.4	729.2
		lcm/hr	948.0	948.0	619.3	619.3	619.3	948.0
(TPHM)		tph	858.3	858.3	1,893.3	958.2	958.2	229.3
(SS)	Scheduled Shifts / Year (from above)	shifts/year	333	333	665	665	665	665
(PH)	Production Hours / Year (from above)	hrs	2,106	2,106	4,739	4,476	4,476	4,332
(TA)	Truck Availability to Shovel	%	95.0%	95.0%	95.0%	95.0%	95.0%	95.0%
(TPHA)	Production Adjusted = TPHM x TA	tph	815	815	1,799	910	910	218
(RP)	Real Production = TPHA x PH	t/year	1,717,468	1,717,468	8,524,196	4,074,479	4,074,479	943,471
	Production / Scheduled Shift = RP / SS	t/shift	5,165	5,165	12,818	6,127	6,127	1,419
	Production / Scheduled Work Hours = RP / WH	tph	652	652	1,619	774	774	174
	Production / Scheduled Production Hours = RP / PH	tph	815	815	1,799	910	910	218

Open Pit Truck Productivity

The design assumptions used to determine hauling productivity are shown in Table 16.1.18.3.

Table 16.1.18.3: Open Pit Hauling Parameters

Schedule Data			Waste 105t	Ore 105t
	Calendar Days	days/year	365	365
	Scheduled Shutdown - weather	days/year		
	Unscheduled Days Down - weather	days/year	15	15
	Mine Work Days	days/year	350	350
	Work Days / Week	days/year	7	7
	Shifts / Day	shifts/day	2	2
	Shifts / Week	shifts/week	14	14
	Scheduled Weeks / Year	weeks/year	50	50
	Shifts / Year	shifts/year	700	700
	Scheduled Hours / Shift	hours/shift	10.5	10.5
	Scheduled Hours / Year	hours/year	7,350	7,350
(T)	**Total Theoretical**	hours/year	8,760	8,760
(SU)	Scheduled & Unscheduled Shutdown	hours/year	1,410	1,410
Standby				
	Lunch Break	hours/shift	0.5	0.5
	Shift Start / Shutdown	hours/shift	0.4	0.4
	Coffee Breaks	hours/shift	0.5	0.5
	Miscellaneous - Blasting & Moves	hours/shift	0.3	0.3
	Total Standby	**hours/shift**	**1.4**	**1.4**
(S)	**Total Standby**	**hours/year**	**980**	**980**
	Available Working Hours	hours/day	21.2	21.2
	Available Working Hours	hours/year	6,370	6,370
Annual Hours				
(T)	**Total Theoretical**	**hours/year**	**8,760**	**8,760**
(S)	**Total Standby**	**hours/year**	**1,410**	**1,410**
(SU)	Scheduled & Unscheduled Shutdown	hours/year	980	980
(W)+(R)	Work + Repair = (T-S-SU)	hours/year	6,370	6,370
(W)	Work = MA x (T-S-SU)	hours/year	4,778	4,778
Mechanical Availability				
	Scheduled Downtime	shifts/year	70	70
	Scheduled Downtime	hours/year	840	840
	Scheduled Downtime		10.0%	10.0%
	Unscheduled Downtime		15.0%	15.0%
	Total Downtime		**25.0%**	**25.0%**
	Shifts Available for Scheduling	shifts	630	630
(MA)	Mechanical Availability		75.0%	75.0%
Physical Availability				
(PA)	Physical Availability = (W+S)/T		65.7%	65.7%
Use of Availability				
(UA)	Use of Availability = W/(W+S)		83.0%	83.0%
Effective Utilization				
(EU)	Effective Utilization = PA x UA		54.5%	54.5%
(WH)	Work Hours / Year	hours/year	4,778	4,778
	Operating Efficiency - operation based		85.0%	85.0%
(PH)	Production Hours / Year	hours/year	4,061	4,061

16.1.19 Open Pit Mine Equipment

The open pit will be mined using a new fleet of equipment except for the Komatsu 105t haul trucks, the water truck and the fuel-lube truck. The open pit equipment required is shown in Table 16.1.19.1.

Table 16.1.19.1: Mine Equipment

	Size	Model	Units
Open Pit Mine			
Rotary Drill	165mm	DM45	1
Hydraulic Drill	110mm to165mm	CM785	1
Mass Excavator	$4m^3$	Cat 365CL	1
Mass Excavator	$10m^3$	O&K RH 90	1
Front End Loader	$6.5m^3$	Cat 988HL	1
Truck	100t	Komatsu 785-3	5
Truck	100t	Komatsu 785-3	1
Bucket (spare)	$10m^3$	O&K	1
Dozer - Track	310hp	Cat D8T	1
Dozer - Track	410hp	Cat D9T	1
Grader	16ft	Cat 16H	1
Water Truck	5k-gal	Cat 725	1
Fuel Lube Truck	1k-gal	L8000	1
Mechanic Truck	-	Sterling	1
Welding/Crane Truck	-	Sterling	1
Pickup Truck	3/4t		15
Light Plant	-	Magnum MTL3060	4
Sanding/Stemming Truck	-		1
Forklift for Warehouse	-	Cat TL1255	1
Backhoe Loader	97hp	Cat 430E Tool Carrier	1
Flatbed Truck	-		1
Crew Vans	-		1
Skid Loader	-	Cat 272C	1
Forklift for Mill		Load Lifter 2414	1
ATV	-		1

Open Pit Water Management

Based on the 1:25 year storm event modeling, the ultimate open pit area could expect to see approximately $33,000m^3$ of water in a 24hr period. The open pit sump design allows for this quantity with the pump system designed to pump this volume over a 48hr period. There will be two pumps (xxkw) that feed two 20.3cm discharge water lines that report to the surface treatment ponds. Normal operations will require a single pump and line to be used. The twin lines allow for routine maintenance and maximum design pumping as required.

16.1.20 Overburden Slopes, Waste and Overburden Stockpiles

AMEC (2008b) carried out a Feasibility Study which included the stability assessment of the following components: overburden slopes around the open pit, clean waste rock stockpile, dirty waste rock stockpile and the overburden stockpile.

Overburden Slopes of Open Pit

In consideration of the nature of the overburden materials and the consequences in the event of a potential large scale failure (specifically potential for loss of life and mine equipment), the overburden slopes of the Open Pit are designed with respect to long-term stability (Factor of Safety ≥1.5) and seismic loading conditions. The following loading cases have been analyzed:

- Steady State Static Loading: Long-term static loading condition; using drained shear strength parameters for granular soil deposits and peak undrained shear strength for the cohesive deposits; and

- Post Earthquake Conditions: Temporary weakening of susceptible silt layer due to partial liquefaction or excess pore pressure generation (use of residual strength parameters, where applicable).

Based on the stability assessments, a slope inclination of 3H:1V is considered adequate for the individual slope benches, which should not exceed 8m in height. The recommended overall inclination for the pit perimeter slope is 5H:1V. Also note the following conditions that should be respected to maintain safe slopes:

- No water ponding is to be permitted at the slope crest surfaces (through satisfactory drainage control); and
- The pit overburden slope should be provided with erosion protection as the slope safety could be compromised by erosion, which could be extensive in the silty and fine-grained sand materials.

Dirty Waste Rock Stockpile

Geochemical investigations of waste rock were carried out by AMEC in conjunction with the geotechnical investigations and design studies. Tasks completed included geochemical characterization of 26 specimens of existing waste rock collected from the surface, 132 core samples from the ore zone, and 80 core samples of waste rock selected from the open pit drilling program. Static testing for selected samples from the various sources was conducted including analyses for: ABA (acid base accounting), concentration of total metals, and British Columbia Ministry of Energy and Mines (BC MEM) leachate extraction. Semi-quantitative mineralogical analyses of some samples were also conducted by Rietveld X-ray diffraction (XRD):

- Most waste rock at the Black Fox property will be net non-acid generating with a safety factor much greater than the conservative screening criterion of 4:1 for the neutralization potential (NP) to acid generation potential (AP) ratio (NP/AP or NPR) for waste rock (Price, 1997). Values for neutralizing potential (determined by analytical titration) are in excess of 100kg aCO_3/t; and
- All waste rock types identified on the Black Fox Property contain elevated concentrations of total arsenic (As), nickel (Ni) and chromium (Cr) relative to average crustal abundances, however, the leachability of these elements is variable and dependent on rock type.

Preliminary analyses conducted by AMEC suggested that the potential exists for leaching of both arsenic and nickel from waste rock produced at the Black Fox Mine. The rock types with such potential will be stored in the Dirty Waste Rock Stockpile. Some temporary storage may be required to account for scheduling lag between waste rock production and use for construction.

The proposed dirty waste rock stockpile will be located south of the open pit, with a rock outcrop hill to the east and a beaver pond to the west. The ground surface on the west side continues to slope down to Lawler Lake. The ground surface elevation within the proposed footprint area of the dirty waste rock stockpile is generally sloping towards west from about elevation 298 to about elevation 292.

The total quantity of dirty waste rock that will be excavated from the Open Pit development is estimated to be about 43.1Mt. The footprint area of the stockpile is roughly 55ha. The stockpile top is at elevation 358. The overall exterior slope of the stockpile, which is based on slope stability analysis results, is about 2H:1V (comprising 10 to 12 m high benches with minimum 10m bench width).

Clean Waste Rock Stockpile

The total quantity of the clean waste rock that will be produced is estimated to be about 2.92Mt, or 1.54Mm^3 (based on 1.90t/m^3 dry density for the dumped material). The site designated for the temporary clean waste rock stockpile is located east of the Open Pit. The existing ground surface of the generally flat lying area is at about elevation 293.

Overburden Stockpile

The total quantity of overburden to be excavated from the Open Pit area is estimated to be about 10.8Mt. The overburden material at the open pit area is variable. The most predominant deposits at the open pit site are the silty sand to sandy silt, silt and till deposits, with some silty clay layers on the east side. The upper part of the silty sand to sandy silt stratum is mostly drained, although the level is relatively high on the east side within the silty clay deposit. This suggests that except for the saturated silty clay deposit, the other deposits will be suitable for excavating, handling and stockpiling.

The overburden material will be stocked in three areas referred to as the East Overburden Stockpile, the North Overburden Stockpile (located north of Highway 101), and the West Overburden Stockpile (located west of Froome Lake and south of Highway 101).

Table 16.1.20.1 presents estimated stockpile capacities and the design features of the three stockpiles. The stockpile capacities take into consideration the volume that will have to be stocked prior to the dam construction and the volume that will be used for dam construction.

Table 16.1.20.1: Principal Features of Overburden Stockpiles

Design Item	East Overburden Stockpile	North Overburden Stockpile	West Overburden Stockpile
Design storage capacity of the stockpile	0.69 x 10^{6}m^3	0.98 x 10^{6}m^3	4.96 x 10^{6}m^3
Estimated final (long-term) storage volume	0.23 x 10^{6}m^3	0.27 x 10^{6}m^3	4.96 x 10^{6}m^3
Footprint area of stockpile	10ha	21 ha	42ha
Existing ground surface elevation	299 to 300	290 to 297	289 to 294
Final elevation of the top of stockpile	302	294	310
Maximum height (initial or final stage)	14 to 16m	2 m (northeast) to 9 m (northwest)	20m
Perimeter slope (H:V) based on slope stability analyses	6H:1V	9H:1V (north, west, and south) to 4:1 (east)	6H:1V (north, west, and south) to 9:1 (east)

16.2 Underground Mining

To continue mining of the ore resources at Black Fox below the bottom of the designed open pit (185m below surface), underground mining methods were reviewed that will minimize dilution, capital and operating costs, maximize recovery of the ore resources while maintaining the design production capacity of the mill.

The recently purchased Stock Mill will be used at a production capacity of 1,100tpd and the remaining production will be shipped to the Holt Mill under a toll milling arrangement. Apollo and SRK developed an operating plan with the open pit and underground mines operating concurrently, with an ore production rate of 1,500tpd from the open pit and 1,000tpd from the underground mine. Offsite milling will be used to process the ore produced for the current ore reserves.

16.2.1 Selection of Mining Method

The medium dipping (45º to 50º) orebody and the relatively complex and nuggety nature of the ore requires the selection of a cut and fill mining method. Alternative underground mining methods, such as longhole stoping, are not applicable due to the potential for greater dilution and ground control issues in shallow dipping stopes.

Cut-off Grade Calculation

The cut-off grade uses a preliminary estimate of the economics of the orebody to determine the portion of the resource above which mining will be profitable. Table 16.2.1.1 presents the input values and the calculation.

Table 16.2.1.1: Underground Mining Cut-off Grade

Parameter	Unit	Amount
Mining cost	$/t	49.00
Milling cost	$/t	13.00
Admin Cost	$/t	4.00
Total Cost	**$/t**	**66.00**
Gold price	$/oz	650
Mill recovery	%	95
Refinery Charges	$/oz	2.50
Net oz value	$/oz	648
Cut-off Grade	gpt	3.3

The mining, milling and administration costs are directly from the Black Fox Prefeasibility Study (SRK, 2007). A cut-off grade of 3gpt was used to filter the block model to determine the economic portion of the indicated resource.

Geotechnical Design Parameters

The underground portion of the Black Fox Project is expected to extend from below the existing mine (near 200m depth below surface) to approximately 500m depth below surface. Geotechnical data has been limited to the logging completed be Apollo Gold staff and additional investigations are recommended to be completed during mine development to confirm assumptions made in this study.

The dataset used to determine the geotechnical properties contained twenty-one drillholes that pierced the ore body, hangingwall and footwall rock at the depth of the planned underground mine. The lithologies were grouped into three main types or domains, Mafic Volcanics (MVs) and Ultramafic Volcanics (UMVs) and faulted zones.

The Rock Mass Rating System (RMR) was utilized to identify the rock characteristics. Three categories for the rock masses were assessed using the RMR system; Fresh Rock, Altered Rock and Faulted Rock. These categories were considered typical of the conditions expected within the ore body. In summary, 75% of the rock mass within the underground mine is expected to be of Fair to Good quality.

Support will primarily be a function of the span, rock mass strength and characteristics, and presence of adverse geologic structure. RMR values identified from the analysis of the drillhole data can provide an estimate of span for man entry mining. The Span Design Curve defines the largest diameter circle, which can be drawn within the boundaries of the exposed back and hangingwall. The assessment determined that the cut and fill stope areas would be stable at a maximum 6m span.

Support within these areas is estimated to require light gauge mesh and "Swellex" or similar type bolts, 1.8m long and 1.2m x 1.2m in the back where necessary. Additional support such as #6 grouted rebar, 1.8m long and 1.2m x 1.2m spacing in back and walls with heavy gauge mesh and 50-100mm of shotcrete may be required in isolated areas where poor or faulted rock mass conditions are encountered. It should be noted that the state of stress within the proposed mine is currently unknown and will need to be investigated further in order to insure the critical span is applicable.

A structural stability analysis was undertaken in order to determine the stability of the development headings. It was found that the 5m x 5m tunnel dimensions, that are the maximum size used in the design, are stable with a standard support pattern.

Stope Design Parameters (Drift and Fill)

The cut and fill design is based on 3m high stopes. The 3m height was chosen primarily to reduce dilution and to improve ground conditions. The Black Fox orebody width varies considerably over the property and designed stopes range between 4m and 35m wide. Where stope widths exceed 6m horizontally, it will be necessary to extract multiple side-by-side drifts on each cut in order to limit the mining span. Figure 16-5 shows an example of the mining sequence for a typical wide cut and fill lift.

Stope sequencing is based on the following design parameters:

- Three stopes consisting of six cuts each are mined as a 54m high block from the bottom up (Figure 16-6). This is a general design guideline and varies across the orebody as a result of changing geometry;
- Once the three stope unit is complete mining moves down to the next three stope block directly below the first;
- The last cut of a block that has been mined above will be a 3m sill pillar. Where the stopes are less than 6m wide and a single drift will be driven, the pillar will be mined in 20m sections on retreat using uppers drilled from the cut below. The 20m sections will be unsupported and will be mucked out using remote LHD's. The 6m high final stope can be left open or loosely filled with uncemented development waste;
- In a sill pillar extraction where the ore is wide and multiple drifts are required within a cut (Figure 16-5) it will be necessary to put 7% binder (50% cement/50% flyash) into the Cemented Rock Fill (CRF) in the first cut of the block above the sill pillar. This high binder content coupled with direct supervision of the placement process will enable the final sill cut to be mined as a standard cut with fill in the back; and
- Five stopes will be mined in order to ensure a consistent ore flow. When mining ore, each of these stopes produces 360tpd. This would produce a total of 1,800tpd for the underground. However, the stope crews are also responsible for: developing the initial

3m x 3m stope access ramp; developing the waste backslash cuts; and backfilling the stopes. Incorporating the waste development and backfill into the schedule brings the average stope tonnage down to 1,000tpd.

Backfill Design

The primary consideration for backfilling is to achieve a consistent tight fill throughout the wider cut and fill areas. This is necessary where multiple drifts will be mined adjacent to each other on the same cut. If tight fill is not achieved the apparent span could get very large resulting in a potential for stope back in stability.

In order to achieve consistent tight fill the mine design will utilize a CRF (3% binder by weight) backfill methodology. The CRF is packed into the stope using a 1.5yd^3 LHD fitted with a rammer to ensure a completely tight fill through the length of the stope.

Waste material is sized on surface, mixed with binder (50% cement/50% flyash) and back-hauled to stopes using the haul trucks. Placement will be at an average rate of 650tpd.

Development Design Parameters

All underground waste development drifts and ramps will be mined at a nominal 4m high x 4m wide. This will allow the largest piece of equipment (27t trucks) to access all areas of the mine up to the stope entrances. A new surface access ramp will be driven at 5m x 5m to take over from the current access ramp that will be eliminated by the open pit. Underground internal ramps will be driven in the footwall of the orebody to access the cut and fill stopes. The internal ramps are developed approximately 70m beyond the ore zone in the footwall.

The cut and fill access ramps are driven at 3m high x 3m wide to match the cut height and because trucks are not required to access the stopes.

The new surface access portal and ramp (5m high x 5m wide) is designed to limit curves and turns to promote efficient truck haulage, and reduce ventilation constraints. The ramp starts close to the surface ore stockpile area to limit underground trucks driving on surface roads, which could be problematic during the winter months.

Interaction between Underground and Open Pit Mining

The open pit will mine out the current underground access portal and ramp at the end of 2010. It will be necessary to have the new surface access ramp complete by this time. The underground mining will extract the orebody in a top down sequence. Open pit mining and operational underground stoping are never in proximity to each other, thus the question of interaction from a blasting and ground control point of view does not arise. There will be a hydrological connection in that any of the surface water reporting to the open pit mine operations and not pumped out will ultimately report to the underground mine operations. Upgrades to the underground water pumping system have been incorporated into the underground mine design.

The open pit benches will be mining through old backfilled and open stopes in the upper levels of the old mine. The process for confirming the location of these stopes through test drilling as the surface mining approaches is discussed in Section 16.1.

16.2.2 Stope Design

The underground stopes were designed by slicing the resource model on 3m plan views. The design was constructed on only the indicated resource blocks above the 3gpt cut-off grade. The

1gpt to 2gpt indicated blocks were displayed during the design process but were only included in the design if higher grade material could be added by including them. The inferred blocks were not displayed during the design process. Polygons were digitized to define the stope outlines on the 3m plan views. The polygons were then extruded 3m vertically to define the 3D stope shape. Figure 16-7 shows an example of a plan view for a typical stope with the stope outline and the indicated resource blocks.

16.2.3 Development Design

The underground development is based on the stope design work described above.

Surface Access Development

Figure 16-8 shows the location of the current surface access ramp through the historic workings. The ramp was driven at a nominal 5m x 5m with a gradient between 10% and 20%. The ramp is in good condition and the schedule is planned to commence mining operations using this access until the open pit progresses to a position where it mines out the portal and the top of the ramp.

Figure 16-9 shows views of the new surface access ramp that will be developed from a new portal site on the south side of the pit, close to the surface ore stockpile area. The new ramp will be developed with a cross section of 5m x 5m and at a -15% gradient. Remucks 15m deep will be driven at 100m spacing down the ramp to facilitate the development mucking process. As the development progresses these remucks will be converted for use as sumps, transformer bays and storage areas. At a point approximately 900m down the ramp a crosscut will be driven to intersect the East Surface Ventilation Raise (downcast), which will be developed by raisebore from surface.

Internal Ramps

Two internal ramps are positioned approximately 70m in the footwall of the ore zone in order to allow access to six cuts per stope. The ramps follow the average dip and plunge of the east and west stope areas.

Ventilation Drifts

Fresh air is forced into the mine through the two new ventilation raises from the surface to 235L and subsequently the east and west sides of the mine. The air flows through the east and west vent crosscuts to the east and west internal vent raise systems. The 3m x 3m internal vent raises feed the fresh air down to the stopes. In the stopes the air is drawn out of the raises with auxiliary fans and ducted to the working face. Exhaust return air travels back up the internal ramp systems, and then flows from the two internal ramps meet at the current access ramp above 235L. The exhaust then travels down to 235L and up the new Surface Access Ramp to the surface.

Stope Access Development

Figure 16-6 presents a typical stope access development layout. There are approximately sixty stope access drifts between the east and west internal ramps and while the design of each is by no means identical, they are all designed with the following general criteria:

- Internal ramps 70m in the footwall of the ore zone;
- Ventilation raises 50m in the footwall of the ore zone;

- 4m x 4m stope access drifts driven off the internal ramp to the ore access ramp;
- 3m x 3m ventilation access drifts driven to connect to internal ventilation raises;
- 4m wide x 5m high remuck/loadout drifts driven on the access side of the ventilation raise to reduce potential dust problems;
- 3m x 3m ore access ramps driven at -15% to access the first cut of the stope; and
- 3m x 3m cut access backslashing, ramping up at +15% to access subsequent cuts.

16.2.4 Production Schedule

The underground mine production schedule was prepared using Minemax iGannt software linked directly to the Gemcom mine design software.

Development Productivity

Table 16.2.4.1 shows the development design parameters used in the preparation of the schedule.

Table 16.2.4.1: Development Scheduling Design Parameters

Development Activity	Size (m)	Rate/Crew* (m/month)
New surface ramp	5 x 5	200
Internal ramp	4 x 4	200
Stope access	4 x 4	200
Stope loadout	4 x 5	200
Vent access drift	3 x 3	180
Stope access ramp	3 x 3	180
Stope cut access backslash	3 x 3	360
Surface raisebore	4m dia.	120
Internal vent raise	3 x 3	120

* Assumes a crew comprises 2 x 10hr shifts per day

The rate of 200m/month for the larger headings is based on two ten-hour shifts per day, cycling a round per shift. The relatively short advance length of 3.3m used in the schedule, compared to the planned 3.7m average advance per round, is used to account for the lost blasts that are a reality in underground mining. The smaller drifts are planned at a rate of 180m/month or 3m/shift using the same logic.

Stope cut access "backslashing" will advance at a rate of 360m/month with these backslash areas mined by "uppers" out of the preceding stope access ramp, rather than as advancing faces. This process will double the rate of advance due to the fact that drilling, loading and ground support can now be continuous activities throughout the shift instead of cycling between each activity during a shift.

Alimak and raisebore contract raise development is based on a 2m face advance per shift, this allows for mobilization and de-mobilization time at each location.

Stoping Productivity

It was assumed during the preparation of the stoping schedule that a stope crew would be responsible for: developing the 3m x 3m stope access ramps and backslash cuts in waste; carrying out the stope ore mining; and backfilling the stopes with CRF.

Development productivity for the stope access ramp and the stope cut "backslashing" is as presented in Table 16.2.4.1. During ore mining each stope will produce at a rate of 360tpd, with the mine producing 1,000tpd on average. The 360tpd rate is based on two 10 hour shifts cycling a round per shift in a 6m wide x 3m high stope advancing 3.6m per round. CRF is placed at an average rate of 650tpd as required.

Scheduling Process

The design and scheduling process does not address the detail within each cut and fill stope cut, instead it is assumed that as soon as the stope access ramps reach the ore, production will commence at a rate of 360tpd until the complete cut tonnage was mined out. When the cut ore is mined out, backfill is placed at a rate of 650tpd until the stope is filled. When the cut is filled the stope access ramp backslashing for the next cut starts. Over the quarterly timeframe used for the representation of the scheduling results in the economic model, this lack of in-stope scheduling detail is acceptable.

The schedule is built up by linking together stopes to mimic a crew moving from stope to stope over time. Because each stope produces development waste and requires backfill, in addition to mining ore, each area on average produces only 200tpd of ore. It was therefore necessary to schedule five areas mining together to produce 1,000tpd of ore consistently over the life of the mine. It is anticipated that only four mining crews (combined production and development) per shift will be required to keep these five areas producing consistently.

Internal ramp, stope access and raise development is scheduled to follow the progression of the stope mining.

Development and Production Schedule

During the startup production period between August 2008 and February 2009, the development priority will be the development of the new ventilation circuit. Two underground development crews will produce approximately 400m/month during this period. In addition to the development work, there will be some stoping in the areas above 235L that will be mined from the current access ramp.

Tables 16.2.4.2 to 16.2.4.4 present the development and stoping schedule by month for the life of the mine. The new surface access ramp (5m x 5m) starts in March 2009 and mines at a rate of 200m/month to break through into 235L in December 2009. This ramp is scheduled to be completed by the end of 2009 because the current surface access ramp will be mined out by the open pit in early 2010.

The 400m/month mining rate for the ventilation system drifts and the internal ramps (Underground drift/ramp 4m x 4m) drops to an average of 200m/month from February 2009. By this time the drifts for the ventilation system are substantially complete and the focus is on the internal ramps and stope access drift preparation. The manpower from this development reduction will be moved to cut and fill production stoping. In 2010 the production rate on the internal ramps and stope access drifts reduces to a steady state of 100m/month until it is completed in mid-2014. The Underground drift (4m x 5m) item on the tables relates to the remucks (loadouts) developed off each stope access drift by the development crews.

The underground drift/ramp (3m x 3m) relates to the development of the ore access ramps into the ore. This work is carried out by the stope crews in addition to the backslashing, cut and fill ore mining and the backfilling.

Internal 3m x 3m Alimak raise development is as required to support the deepening of the internal ramps and stope blocks.

Table 16.2.4.2: Underground Production Schedule (2008-2010)

Activity	Unit	Aug-08	Sep-08	Oct-08	Nov-08	Dec-08
Underground drift/ramp (5m x 5m)	m	-	-	-	-	-
Underground drift/ramp (4m x 5m)	m	-	-	-	-	-
Underground drift/ramp (4m x 4m)	m	401	380	366	374	327
Underground drift/ramp (3m x 3m)	m	112	33	19	91	60
Stope backslashing	m	26	70	116	56	121
Raisebore (4m dia.)	m	-	87	124	25	99
Alimak (3m x 3m)	m	-	-	-	29	13
Waste	t	21,326	19,724	15,847	17,456	18,531
Ore	t	2,162	3,956	3,854	2,861	5,624
Grade	g/t	12.63	17.64	8.46	6.14	7.56
Gold	oz	878	2,244	1,049	565	1,367
Backfill	t	1,385	3,161	3,087	1,640	3,284

Activity	Unit	Jan-09	Feb-09	Mar-09	Apr-09	May-09	Jun-09	Jul-09	Aug-09	Sep-09	Oct-09	Nov-09	Dec-09
Underground drift/ramp (5m x 5m)	m	-	-	200	197	203	197	203	203	197	203	197	72
Underground drift/ramp (4m x 5m)	m	-	1	75	30	-	15	43	17	-	15	30	128
Underground drift/ramp (4m x 4m)	m	406	185	178	314	203	147	123	167	158	71	171	57
Underground drift/ramp (3m x 3m)	m	119	116	391	151	112	202	169	31	-	85	29	233
Stope backslashing	m	-	54	69	90	247	147	119	233	304	285	288	103
Raisebore (4m dia.)	m	124	17	-	-	-	-	-	-	-	-	-	-
Alimak (3m x 3m)	m	-	-	-	91	54	-	56	-	-	32	41	128
Waste	t	24,150	12,243	31,287	31,661	27,493	23,167	24,256	21,945	20,863	21,576	25,409	23,622
Ore	t	1,514	2,764	7,693	16,984	20,517	15,969	25,920	20,111	28,904	16,598	23,858	21,347
Grade	g/t	6.22	3.72	5.65	10.20	15.95	12.11	7.54	7.05	8.03	8.45	7.61	9.30
Gold	oz	303	330	1,398	5,569	10,520	6,217	6,280	4,561	7,459	4,509	5,840	6,380
Backfill	t	1,159	1,391	4,372	11,320	13,335	10,440	10,621	21,326	9,572	19,795	15,173	14,516

Activity	Unit	Jan-10	Feb-10	Mar-10	Apr-10	May-10	Jun-10	Jul-10	Aug-10	Sep-10	Oct-10	Nov-10	Dec-10
Underground drift/ramp (5m x 5m)	m	-	-	-	-	-	-	-	-	-	-	-	-
Underground drift/ramp (4m x 5m)	m	-	-	-	27	15	-	-	31	15	30	15	16
Underground drift/ramp (4m x 4m)	m	102	92	80	82	161	97	46	70	105	116	58	63
Underground drift/ramp (3m x 3m)	m	350	-	49	80	121	102	91	162	123	325	79	114
Stope backslashing	m	142	269	401	268	178	270	174	162	274	121	273	385
Raisebore (4m dia.)	m	-	-	-	-	-	-	-	-	-	-	-	-
Alimak (3m x 3m)	m	-	-	35	-	-	-	40	-	-	-	-	55
Waste	t	13,977	8,628	13,182	12,447	14,434	11,445	8,788	10,981	13,520	16,961	9,802	15,979
Ore	t	22,350	29,005	26,075	26,539	27,963	27,766	28,746	27,101	25,133	19,601	31,490	21,907
Grade	g/t	8.19	10.17	9.21	10.33	13.30	12.26	12.89	12.59	12.50	10.53	10.60	12.52
Gold	oz	5,888	9,487	7,723	8,814	11,959	10,944	11,916	10,969	10,100	6,633	10,734	8,815
Backfill	t	12,972	18,465	21,553	21,330	22,563	16,991	20,778	16,072	21,276	13,236	16,259	21,501

Table 16.2.4.3: Underground Production Schedule (2011-2013)

Activity	Unit	Jan-11	Feb-11	Mar-11	Apr-11	May-11	Jun-11	Jul-11	Aug-11	Sep-11	Oct-11	Nov-11	Dec-11
Underground drift/ramp (5m x 5m)	m	-	-	-	-	-	-	-	-	-	-	-	-
Underground drift/ramp (4m x 5m)	m	31	-	30	-	-	-	11	19	15	15	15	-
Underground drift/ramp (4m x 4m)	m	26	127	117	98	102	98	138	60	83	95	66	60
Underground drift/ramp (3m x 3m)	m	242	7	193	104	83	-	11	114	21	93	195	164
Stope backslashing	m	273	245	184	231	383	282	189	226	202	221	206	72
Raisebore (4m dia.)	m	-	-	-	-	-	-	-	-	-	-	-	-
Alimak (3m x 3m)	m	-	-	-	-	-	-	-	85	49	-	-	-
Waste	t	14,826	11,257	15,327	10,510	12,059	10,482	10,399	12,519	9,760	10,540	12,891	7,980
Ore	t	34,955	25,499	29,200	33,103	38,528	34,416	28,686	24,841	33,076	25,574	18,959	28,659
Grade	g/t	10.25	10.70	11.34	11.55	11.15	12.09	8.24	11.10	18.54	11.27	10.34	10.13
Gold	oz	11,515	8,770	10,648	12,292	13,809	13,381	7,598	8,862	19,714	9,262	6,304	9,334
Backfill	t	17,181	27,201	18,328	24,377	19,440	26,740	21,071	23,439	18,891	21,976	12,659	9,582

Activity	Unit	Jan-12	Feb-12	Mar-12	Apr-12	May-12	Jun-12	Jul-12	Aug-12	Sep-12	Oct-12	Nov-12	Dec-12
Underground drift/ramp (5m x 5m)	m	-	-	-	-	-	-	-	-	-	-	-	-
Underground drift/ramp (4m x 5m)	m	15	16	-	-	-	-	-	29	-	-	-	15
Underground drift/ramp (4m x 4m)	m	70	55	102	98	102	98	77	40	73	102	98	65
Underground drift/ramp (3m x 3m)	m	104	21	74	-	75	9	26	145	79	64	-	104
Stope backslashing	m	120	78	194	37	191	225	146	157	152	51	246	100
Raisebore (4m dia.)	m	-	-	-	-	-	-	-	-	-	-	-	-
Alimak (3m x 3m)	m	-	-	-	-	-	-	-	83	-	-	-	-
Waste	t	8,397	5,276	10,033	5,321	10,073	9,312	6,770	12,222	8,035	7,187	9,325	7,244
Ore	t	28,354	29,156	30,530	30,178	27,726	30,255	33,602	27,799	29,898	20,938	26,616	19,612
Grade	g/t	6.85	7.76	7.85	8.17	7.45	7.72	8.32	8.05	10.04	14.43	12.69	6.48
Gold	oz	6,247	7,274	7,707	7,929	6,644	7,509	8,988	7,197	9,651	9,712	10,859	4,084
Backfill	t	22,736	24,653	19,188	26,255	22,098	20,529	21,848	20,820	21,677	27,126	12,744	21,243

Activity	Unit	Jan-13	Feb-13	Mar-13	Apr-13	May-13	Jun-13	Jul-13	Aug-13	Sep-13	Oct-13	Nov-13	Dec-13
Underground drift/ramp (5m x 5m)	m	-	-	-	-	-	-	-	-	-	-	-	-
Underground drift/ramp (4m x 5m)	m	17	12	15	15	15	-	16	15	15	15	15	-
Underground drift/ramp (4m x 4m)	m	51	80	76	67	21	22	75	59	68	87	69	102
Underground drift/ramp (3m x 3m)	m	106	-	92	93	244	186	10	25	53	10	13	91
Stope backslashing	m	254	193	141	47	52	92	109	226	54	372	56	103
Raisebore (4m dia.)	m	-	-	-	-	-	-	-	-	-	-	-	-
Alimak (3m x 3m)	m	32	70	-	-	-	-	-	-	-	-	-	-
Waste	t	11,599	9,933	9,101	6,800	7,436	6,590	6,245	8,719	6,240	10,645	5,081	7,470
Ore	t	31,257	32,458	22,842	22,780	32,053	31,046	34,583	30,189	31,382	26,164	38,905	23,906
Grade	g/t	5.40	8.24	6.95	6.09	7.06	6.59	5.31	4.17	4.96	6.26	6.85	5.62
Gold	oz	5,431	8,601	5,105	4,462	7,270	6,576	5,906	4,046	5,003	5,268	8,573	4,321
Backfill	t	16,372	15,835	26,184	19,276	18,436	21,753	22,257	23,203	22,107	24,997	16,947	28,800

Table 16.2.4.4: Underground Production Schedule (2014-2016)

Activity	Unit	Jan-14	Feb-14	Mar-14	Apr-14	May-14	Jun-14	Jul-14	Aug-14	Sep-14	Oct-14	Nov-14	Dec-14
Underground drift/ramp (5m x 5m)	m	-	-	-	-	-	-	-	-	-	-	-	-
Underground drift/ramp (4m x 5m)	m	-	-	15	30	-	-	15	15	-	-	-	-
Underground drift/ramp (4m x 4m)	m	102	92	71	47	102	98	75	17	-	-	-	-
Underground drift/ramp (3m x 3m)	m	117	52	262	116	21	2	163	85	56	-	54	69
Stope backslashing	m	312	213	261	220	223	152	88	263	279	234	52	239
Raisebore (4m dia.)	m	-	-	-	-	-	-	-	-	-	-	-	-
Alimak (3m x 3m)	m	-	-	49	-	-	-	78	-	-	-	-	-
Waste	t	11,561	9,396	15,221	9,995	9,065	6,917	11,015	8,431	6,253	4,377	2,268	6,111
Ore	t	40,071	33,338	25,056	38,428	31,083	31,205	25,234	35,632	18,563	25,732	27,543	26,393
Grade	g/t	5.97	5.38	5.51	7.01	6.64	8.57	6.43	6.90	8.01	9.59	6.90	5.12
Gold	oz	7,691	5,770	4,434	8,662	6,639	8,602	5,219	7,902	4,779	7,932	6,113	4,343
Backfill	t	17,246	22,286	25,678	19,394	22,925	22,643	23,193	12,791	29,211	16,786	13,360	18,153

Activity	Unit	Jan-15	Feb-15	Mar-15	Apr-15	May-15	Jun-15	Jul-15	Aug-15	Sep-15	Oct-15	Nov-15	Dec-15
Underground drift/ramp (5m x 5m)	m	-	-	-	-	-	-	-	-	-	-	-	-
Underground drift/ramp (4m x 5m)	m	-	-	-	-	-	-	-	-	-	-	-	-
Underground drift/ramp (4m x 4m)	m	-	-	-	-	-	-	-	-	-	-	-	-
Underground drift/ramp (3m x 3m)	m	-	66	-	77	56	78	-	-	50	18	-	-
Stope backslashing	m	230	213	85	52	206	194	115	101	-	199	54	-
Raisebore (4m dia.)	m	-	-	-	-	-	-	-	-	-	-	-	-
Alimak (3m x 3m)	m	-	-	-	-	-	-	-	-	-	-	-	-
Waste	t	4,276	5,528	1,240	2,991	6,345	5,612	2,528	1,784	1,270	3,990	1,024	-
Ore	t	20,362	16,288	25,992	11,878	10,161	16,849	20,726	8,015	7,950	13,424	12,375	633
Grade	g/t	6.59	6.11	13.00	11.14	9.85	6.12	7.34	9.60	8.67	5.93	6.74	6.13
Gold	oz	4,314	3,199	10,861	4,256	3,218	3,316	4,894	2,474	2,216	2,560	2,682	125
Backfill	t	28,200	6,194	7,362	16,395	15,850	7,507	7,930	13,599	11,984	4,244	3,069	11,607

Activity	Unit	Jan-16	Feb-16
Underground drift/ramp (5m x 5m)	m	-	-
Underground drift/ramp (4m x 5m)	m	-	-
Underground drift/ramp (4m x 4m)	m	-	-
Underground drift/ramp (3m x 3m)	m	-	-
Stope backslashing	m	-	-
Raisebore (4m dia.)	m	-	-
Alimak (3m x 3m)	m	-	-
Waste	t	-	-
Ore	t	6,640	2,216
Grade	g/t	7.04	6.01
Gold	oz	1,503	428
Backfill	t	5,221	2,043

16.2.5 Mining Method

Drilling and Blasting

The nominal production heading size will be 3m high x 6m wide. This heading size will allow the engineering design team and the ore control geology group the flexibility to maximize the gold grade from the underground mine. The mining cycle involves drilling, blasting, mucking and ground control cycle and a backfill cycle using CRF.

The critical time path to cycle a heading is the jumbo drilling time requirement. To meet production requirements of the Black Fox mine the nominal 3m high x 6m wide production headings will require a double boom jumbo drill drilling 55 holes that are 45mm diameter and 3.6m deep. There will be one drill operator per jumbo drill with a total of two double boom jumbo drills and one single boom jumbo drill operated per shift to meet ore production requirements of 1,000tpd (an additional 2 boom jumbo drill is required for development). A total of five working faces will be available per shift. The mining schedule was developed based on three ore faces cycled per shift (180t/working face/shift - 1,000tpd).

Once the jumbo drill completes the drilling cycle the ANFO explosive is loaded into the holes with the respective nonel blasting cap and booster. The actual advance per round is 3.3m allowing for the difference between the drill depth and the actual pull depth.

Mucking and Haulage

A 4yd^3 diesel LHD is used to muck out the heading after blasting and clearing, the back and ribs are then check scaled by hand. The LHD trams the ore to the associated muckbay (loadout) for each respective production stope. Once the heading is mucked out the bolting process can begin. When the bolting and drilling cycle has started on the next production cycle in the stope, the ore material previously placed in the muckbay is transferred by the 27t haul trucks to the surface ore crusher facility.

Ground Control

Ground support in the form of 1.8m Swellex bolts will be installed using stopers off the floor of the stope or a leveled muckpile. The bolting pattern will be on a 1.2m x 1.2m with 9 gauge wire mesh for areal coverage. This ground control process will be under constant review by management, engineering staff, safety group, Ministry of Labour and most importantly the miners to ensure a safe working environment.

Backfill Cycle

The underground mining cycle will include backfilling the void remaining after the last round has been extracted for each 3m x 6m drift within a cut. The backfill process will be required to provide a working platform for the next 3m horizontal cut for the stope area. The CRF material will consist of waste material sourced from either underground waste rock development or waste rock from the open pit waste rock stockpile on the surface. A cement/flyash (50%/50%) binder slurry will be added at the rate of 3% (by weight) to the sized (less than 150mm) waste material and back hauled underground using the 27t haul trucks to the respective muckbay/loadout for the stope that is being filled.

The CRF is picked up from the loadout and placed in the stope using a 4.0yd^3 LHD, pushed up as far as practical and then pushed tight with a 1.5yd^3 LHD with a jammer plate attached to the bucket.

There will be certain areas in the underground mine that will require mining underneath previously backfilled stopes. These sill cut areas will require special considerations before and during backfill placement. These considerations will include:

- Increasing binder addition to 7% in the first cut of a stope which will be mined from below;
- Direct supervision of the backfill placement process to ensure tight fill and good compaction;
- Protruding wall reinforcement with longer bolts in the first cut of a stope that will be mined from below. This would reduce the likelihood of shear failure of the CRF roof at the CRF/hangingwall interface; and
- Use of remote controlled mucking equipment during the subsequent mining sequence beneath the previously filled stope.

16.2.6 Underground Mine Ventilation

The existing ventilation circuit consists of two, 150kW ventilation fans that discharge air down the east ventilation shaft to the working areas with the exhaust returning up the current surface access ramp. The current maximum volume is approximately 110m^3/sec, which is more than sufficient during the care and maintenance phase.

The underground mine ventilation circuit design is based on the use of two new surface vent raises (4m diameter raisebores) from surface to 235L. These raises will be used to intake fresh air into the mine with the exhaust returning up the new Surface Access Ramp. The exhaust air will be warm enough that freezing will not occur on the ramp roadway and affect the maneuverability of the haul trucks and ancillary equipment. Each surface intake ventilation raise will be equipped with a propane heater used in the winter months to keep the ventilation shafts free of ice. The main escape way will be the new Surface Access Ramp and the secondary escape ways will be in the East and West Surface Ventilation raises. The vent raises will be equipped with ladders and landings to allow the miners a safe method of egress in fresh air if required.

Table 16.2.6.1 summarizes the ventilation requirements for the mine based on the year 2014 production and development requirements for diesel-powered equipment. Year 2014 was chosen to represent a worst-case scenario when the mine is still at maximum production capacity and the production is sourced from the lower levels. The methodology is conservative and allows for the design and construction of the surface and underground ventilation facilities on a permanent basis.

Table 16.2.6.1: Underground Ventilation Requirements

Item	Units	Unit Rating (kW)	Utilization %	Est Power (kW)	Airflow (m^3/s)	East Side Allocation (m^3/s)	West Side Allocation (m^3/s)
Drill Jumbos – 2 Boom	3	87	50	131	8	5	3
Drill Jumbo – 1 Boom	1	55	50	28	2	0	2
LHD-4yd^3	4	157	100	628	38	19	19
LHD-1.5yd^3	1	70	100	70	4	4	0
Haul Trucks	3	298	100	894	54	36	18
Scissor Lift	2	60	50	60	4	2	2
Road Grader	1	112	50	56	3	2	1
Fuel/Lube Truck	1	112	50	56	3	2	1
Boom truck	1	112	50	56	3	2	1
Tractors	4	24	100	96	6	3	3
Subtotal	***21***	***1,087***		***2,074***	***125***	***75***	***50***
Misc. allowance (20%)					25	15	10
Total					**150**	**90**	**60**

The primary airflow is down the East and West Surface Vent raises to 235L where a series of air doors and auxiliary ventilation fans direct the fresh air to the operating development and production faces. As sections of the mine are phased out and access is no longer required, bulkheads will be installed in strategic areas to eliminate these areas from the ventilation circuit and thereby significantly reduce the future ventilation requirements for these areas.

The ventilation requirements of 150m^3/sec are based on 0.06m^3/s/kW, which meets the Ontario standard. The system will need to be upgraded from the current 110m^3/sec capacity by moving the two existing 150kW fans to the new East SurfaceVent Raise and adding a new 120kW fan to the new West Surface Vent Raise. Additional fans will be required for the internal east and west vent raises as well as the new internal exhaust booster fan.

Figure 16-11 shows the location of the ventilation fans, flow rates and distribution. Smaller stope fans and flexible vent duct is used to direct the ventilation air to the working faces. The return airflow is directed up the access ramps to surface.

The Black Fox underground mine currently heats the ventilation air during the months of December through April with a propane fired shaft heater located on the surface over the existing East Vent Raise. The existing shaft heater has the capacity to deliver 4.1 MW (14.0Mbtu) which is adequate for the new East Vent raise requirements. Once the construction of the new East and West Vent Raises have been completed the ventilation upgrade will include moving the existing 150kW vent fans and shaft heater to the East Vent Raise and installing a new 120kW ventilation fan and new 2.6MW (9.0Mbtu) propane fired shaft heater on the surface over the new West Vent Raise.

16.2.7 Underground Mine Equipment

Mobile

Mobile equipment for the underground mine operation will consist of the following diesel powered equipment.

Table 16.2.7.1: Diesel Powered Equipment

Description	Total	3rd Qtr 2008	4th Qtr 2008	1st Qtr 2009	2nd Qtr 2009
Two Boom Jumbo Drills					
Tamrock H205D (reconditioned)	1	1			
MTI-DR-2SB (new)	2	1	1		
Single Boom Jumbo Drill					
Tamrock – H105M (reconditioned)	1	1			
Diesel 4.0yd^3 LHD					
Wagner ST3.5 (reconditioned)	2	2			
MTI-LT-650 (new)	2		1	1	
Diesel 1.5yd^3 LHD (new)	1		1		
27t Diesel Haul Truck					
Tamrock EJC-430 (reconditioned)	2	2			
MTI-DT-3004 (new)	1				1
Scissor Lift					
Getman A64 (reconditioned)	2	2			
MTI-UVT-SL6 (new)	1				1
Grader (existing)					
ANFO Loader (reconditioned)	2	1			1
Fuel/Lube Truck (reconditioned)	1	1			
Boom Truck (new)	1	1			
Man Carrier MTI-MUT210 (new)	1	1			
Tractors (new)	6	3			3

Fixed capital for the Black Fox underground mine will include facilities on the surface as well as underground. Table 16.2.7.2 details the type of capital and the required timing for the respective item to be operational to meet the previously described production schedule.

Table 16.2.7.2: Fixed Capital Equipment

Description	Q3 - 2008	Q4 - 2008	Q1 - 2009	Q2 - 2009
Surface Facilities				
Ventilation Upgrade		X		X
Cemented Rock Fill – Silo / Mixer Facilities	X			
Compressed Air Station				X
Electrical System Upgrade				X
Underground Facilities				
Water Management				X
Electrical power distribution				X
Ventilation Facilities				X
Underground Shop				X

Staffing

The Black fox Project will require highly trained underground miners, support staff, diesel mechanics, and a technical management group that is familiar with mechanized cut and fill mining techniques. The project is located in an active open pit and underground mining area where such individuals with the required training and education are readily available. Recent demand for such individuals, in the mining field, has dramatically increased with a subsequent substantial increase in wages and benefits that will be paid to attract and retain these experienced individuals. An on-the-job training program will be designed and implemented to ensure that properly trained and experienced miners will continue to be available for the mine operation. A rigorous safety training program will also continue to be implemented and updated in order to create the continuous improvement program requirements for a modern mining operation. Many of the salaried level positions will be shared between the open pit and underground operations to achieve greater efficiencies in the Black Fox operations. These positions include engineering and geology staff, environmental, human resources and safety.

Salaried

The direct management group for the underground operations is summarized in Table 16.2.7.3. An experienced and well-educated management and technical group will be required to complete the final design and implementation of the underground mine development and production plans to meet the underground production requirement of 1,000tpd. Ore grade control will be an integral part of this plan and the grade control/geological group will be an integral part of this team. The salary rates were provided by Apollo and reflect recent salary surveys in the Timmins district.

Table 16.2.7.3: Black Fox Underground Management Summary

Salary Position	Positions	Annual Salary(US$)	Burden (30%)	Total Annual Salary with Burden
Underground Mine Manager	1	120,000	36,000	156,000
General Mine Forman	1	90,000	27,000	117,000
Shift Foreman	4	80,000	24,000	416,000
Maintenance Planner	1	60,000	18,000	78,000
Total	**7**			**US$767,000**

There are no bonus rates assigned to these salary totals at this time. A burden of 30% has been assigned to all positions.

Hourly

The crew schedule will provide operations coverage based on two 10.5hr (paid) shifts per day – 7d/wk. There will be a total of four crews with each working four days on and four days off on a rotation schedule. The effective hours for each shift will be 9.5hrs allowing for lunch, breaks, and travel time. Based on this schedule of 4 days on and 4 days off the total paid hours per year is 1932. Each crew will be made up of a total of 24 workers including a development crew (varies from 3 to 5 workers depending on the project year), production crew (15 workers) and the mechanic group (6 workers). Staffing and job titles are summarized in Table 16.2.7.4 for the maximum hourly staffing requirements. The hourly rates were provided by Apollo and reflect recent salary surveys in the Timmins district. A bonus schedule has been included with an approximate 35% increase over the base rate to allow for the competitive nature of the mining sector at this time. An additional 30% burden rate based on the base rate has been added to account for the taxes and benefits.

Table 16.2.7.4: Hourly Manpower Summary – 1,000tpd (Full Development Crew)

Description	Miners per Crew	Total Miners	Hourly Rate US$ / hr	Annual Pay	Bonus 35 %	Burden 30%	Annual Total
Development							
Miner 1st	2	8	26.35	404,100	141,400	121,200	666,800
Miner 2nd	2	8	25.91	397,400	139,100	119,200	655,600
Helper	1	4	23.00	176,400	61,700	52,900	291,000
Production							
Miner 1st	3	12	26.35	606,200	212,200	181,800	1,000,200
Miner 2nd	2	8	25.91	397,400	139,100	119,200	655,600
Helper	1	4	23.00	176,400	61,700	52,900	291,000
LHD Operator	3	12	25.00	575,100	201,300	172,500	948,900
Truck Driver	3	12	23.00	529,100	185,200	158,700	873,000
Road Maint.	1	4	26.35	202,100	70,700	60,600	333,400
Utility Crew	2	8	25.00	383,400	134,200	115,000	632,600
Mechanics							
Leadman	1	4	29.55	226,600	79,300	68,000	373,900
Mechanic 1st	2	8	26.35	404,100	141,400	121,200	666,800
Mechanic 2nd	2	8	25.91	397,400	139,100	119,200	655,600
Lube/Tire	1	4	25.00	191,700	67,100	57,500	316,300
Total	**26**	**104**					**8,360,700**

16.2.8 Support Services

Compressed Air

Compressed air will be required in the development and production headings to operate the handheld jackleg and stoper drills required for drilling holes for ground control bolts, the Alimak raise contractors and utility requirements throughout the mine. The current compressor equipment will be used to supply the compressed air requirements for the initial phase of the reopening of the Black Fox underground mine.

Water Management

Water management for the Black Fox underground mine was designed to handle approximately 2500m^3/day on average of excess ground water. Most of the new sump and pump design capacity (located on 235L) is due to the excess water reporting from the ultimate open pit and will be designed to handle up to 15,000m^3/day. The underground water system upgrades will be phased with the open pit expansion schedule. The underground headings will have small pumps that pump water to the intermediate sumps located in the new east and west haulage ramps. The intermediate sumps will subsequently pump water to 235L main sump. The drill water sump will supply pressurized water to the headings and other areas of the mine using the drill water distribution system. Excess water from the mine will be pumped to the surface using two 25.4cm (10in) water lines installed in the new East Surface Vent Raise. Normal operating conditions will require the use of one pump and line at a time. The twin system allows for routine maintenance and using both pumps as required during a 1:25 year storm event on the surface. The underground excess water will report to surface water distribution system to be treated and discharged.

Electrical Systems

The new feeder will be installed (5kV 3-250 MCM riser teck cable) in the new East Vent Raise and used to supply a new 5kV transformer station centrally located on 235L. An existing 300kVA portable substation will be relocated on 235L to supply the new water management system (discharge and drill water), lighting, new shop and other local requirements. An additional existing 300kVA portable substation will be used to supply power for the new internal ventilation fans located at the top of the raises once they are completed in the 1st Qtr 2010. The existing 750kVA portable substation will be installed to supply the internal east side power requirements for the jumbo drills, diamond drilling, pumps, stope fans, lighting, etc. The internal west side power requirements will be supplied by relocating an existing 600kVA portable substation. All of the existing electrical equipment will be thoroughly tested and repaired as required before it is reinstalled to ensure safe operation. A temporary 300kVA portable substation will be installed on the surface in the 1st Qtr 2009 to supply power requirements for the new surface access decline. Once this decline is completed to 235L (1st Qtr 2010) the temporary substation will be relocated underground.

Communications

Underground communications will be an important part of managing the safe development and production requirements of the mine operation. The mine currently has hard line underground mine phones located at strategic locations throughout the mine. As the mine expands the mine phone system will be expanded. New phones will be installed at the new permanent underground ventilation and electrical facilities, underground shop and refuge stations as required. The existing leaky feeder communication system will also be upgraded as required to allow better communication within the mine.

Underground Shop

The new underground shop will be used to perform most of the maintenance requirements on the underground equipment fleet. Located on 235L it will provide easy access to the east and west production areas of the mine. Routine tire repairs, oil changes and preventive maintenance procedures will be performed at this facility. Depending on type of equipment and extent of the work required the equipment may be moved to the surface maintenance facility. The underground facility will be equipped with an overhead crane, compressed air, lighting, welding equipment, equipment wash down area, and a small secure warehouse area.

16.2.9 Health and Safety Considerations

Refuge Stations

Refuge stations will be installed in accordance with provincial and national requirements for underground mines. The stations will be constructed in muckbays that are no longer needed at strategic locations around the mine. There will be one in the east side production area and one in the west side. Each station will be supplied with air (from the compressed air line), water, and communication equipment including mine phones and leaky feeder connections.

Mine Rescue

Apollo has prepared a Memorandum of Understanding with the local MASHA authority that establishes the responsibilities for the underground mine. With approximately 25 to 26 miners underground at any one time the mine will require a minimum of 10 trained mine rescue personnel. The mine will use MASHA supplied equipment and training as part of the workplace safety insurance board premiums that are paid by the mine.

In the event of a mine incident that requires the immediate evacuation of the mine Stench gas (methyl mercaptine) will be introduced to the compressed air system as well as the main east and west intake ventilation fans on the surface. Miners will be trained to make their way quickly to the closest refuge station along well marked escape routes.

16.2.10 Surface Facilities for Underground Mine

Office and Dry Requirement

Lockers, change room and shower (Dry) faculties for 100 underground miners has been allowed for in the surface mine facility design and costing. While only 25 miners will be on site at any one time lockers for 100 miners will be required. Separate management dry facilities will be required with the capacity for 20 salaried individuals. Separate women's dry facilities for an estimated five women have also been allowed for. Separate office requirements for shifters, lamp room, safety office, and meeting rooms have also been allowed for in the surface facilities design.

Backfill

The cemented rockfill (CRF) will require the addition of binder added to the sized waste rock stored on the surface. The binder will be a 50% cement and 50% flyash mixture commonly used for this purpose. The binder will be added to the underground haul trucks from a Binder Batcher system where water is mixed with dry cement/flyash and delivered to the haul trucks from an overhead spray nozzle. The silo will have a capacity of $85yd^3$ of dry binder that is delivered as required (estimated monthly usage is 600t of binder). The batch system will be calibrated to deliver the required portion of mixed binder (3% by weight for the CRF). Provisions will be required to heat the water used for mixing cement to prevent freezing during the winter months.

The CRF will require a sized waste rock material to allow for proper mixing with the cement. The nominal size will be less than 150mm material mixed with the fine material. Sizing will be provided by a front-end loader placing run of mine waste from the open pit and/or underground development rock on an angled grizzly arrangement with the required spacing between grizzly bars of 150mm. The over sized material is placed back on the waste rock stockpile and the undersized material is placed on the underground backfill stockpile. As required, the backfill material is loaded on the underground haul trucks and transferred underground after the additional of the binder mixture.

16.3 Tailings Storage Facility at Stock Mill

16.3.1 Tailings Management

The ore from Black Fox site will be partially processed from the Stock Mine mill. Golder Associates Ltd. has recently prepared a pre-feasibility report entitled "Conceptual Design of Phase 4 and 5 Raises, Tailings Management Facility, Stock Mine, Ontario". As requested by Apollo, AMEC has conducted a preliminary review of the design report. As some of the important information required to complete the review is not available currently, the design background data and accuracy of the design study cannot be confirmed. The most significant issues involved include:

- Absence of static liquefaction assessment of tailings, which is a key issue in tailings dams raised by upstream method of construction.
- Lack of base filter drains or filter blanket to control phreatic surface within the dam (thus, the phreatic surface is likely either to emerge on the dam slope surface or be very close to it).
- The design strength values used for the existing and future (improved by preloading) appear to be high (in comparison to the N-values mentioned in the report) for sensitive silty clays that exist in the general area (site specific strength and sensitivity data is not available).

To advance the study to a feasibility level, Apollo has commissioned AMEC to carry out a detailed evaluation of the current concept or develop a new concept for the tailings management at the Stock Mine site and this study is currently in progress.

16.3.2 Water Management

AMEC carried out an overall site water management study for the Black Fox site, with a Holding Pond as the water management centre. The following sections describe in details of the concept and design features.

Design Criteria and Considerations

1. The Holding Pond will serve as the central water management facility for the entire site. Contaminated water from the Open Pit, Underground Operations and Dirty Waste Rock Stockpile will be pumped to the Holding Pond and from there sent to treatment plant for treatment. The Holding Pond was designed to provide sufficient storage capacity for a minimum 5-day retention time for the 1:25-year rainfall storm volume (Environmental Design Flood, (EDF). The pond water will be allowed to spill for flood events exceeding the EDF.

2. Surface runoff from the Open Pit will be continuously pumped to the Holding Pond with a capacity that prevents substantial accumulation of water in the pit (leading to production disruption) for hydrologic events up to 1:25 year event.

3. Runoff from the dirty waste rock stockpile will be collected for temporary storage and pumped to the Holding Pond for treatment. The collection system was designed for the 1:25 year event. The pump was sized to effectively draw down the collection pond to the minimum operating level within 5 days after the design event.

4. No runoff collection or treatment system is required for the clean waste rock stockpile and the overburden stockpile. Surface runoff shall be discharged to the environment with sediment control measures in place, if required.

The overall site water management schematic and water balance under average year runoff condition is shown on Figure 16-12.

Summary of Design Parameters

The principal design features are summarized below.

Open Pit

The required pumping capacity was designed at 33,020m^3/d (about 6,060gpm) to be able to pump the runoff resulting from a 1:25 year 24hr storm together with the seepage from the underground workings. The inflow design parameters used to deriving the required pumping rate are summarized in Table 16.3.2.1.

Table 16.3.2.1: Open Pit Pump Capacity Design Parameters

Description	Parameter
Total footprint area for the Open Pit	33.0ha
The rainfall depth corresponding to the design event (1 in 25 year 24 hr rainfall)	98.3mm
Total runoff volume corresponding to the design event	32,440m^3
Seepage from Open Pit and underground workings to the design duration	580m3
Total flow volume corresponding to the design event	33,020m^3

Dirty Waste Rock Stockpile

The seepage and runoff emerging along the perimeter toe of the Dirty Waste Rock Stockpile will be isolated from the clean runoff from the surrounding areas by a containment berm and will be directed to a pump sump at the low lying area at the northwest corner of the stockpile. The collected water from the sump will be pumped to the Holding Pond. The storage capacity of the pump sump is designed for 1 in 25 year 24 hr rainfall event as in Table 16.3.2.2.

Table 16.3.2.2: Dirty Waste Rock Stockpile Pump Sump Design Parameters

Description	Parameter
Total footprint area for the Dirty Waste Rock Stockpile (within the diversion berm)	55.0ha
Runoff coefficient used for the waste rock stockpile	0.8
The rainfall depth corresponding to the design event (1 in 25 year 24 hr rainfall)	98.3mm
Total runoff volume corresponding to the design event (without pumping)	44,300m^3
Storage requirement corresponding to the design event (with 700gpm constant pumping rate)	40,500m^3

The collection pond would be pumped to minimum operation level in about 5 days with the design pumping rate of 3,800m^3/d (about 700gpm), under the 1:25 year hydrologic event.

Holding Pond

Table 16.3.2.3 summarizes the inflow design parameters used to deriving the required pumping rate and storage capacity in the Holding Pond.

Table 16.3.2.3: Holding Pond Inflow Design Parameters

Description	Annual Volume (m^3/year)	1:25 year 24-hour Volume(m^3/d)
Inflows		
Direct precipitation over the pond surface	109,000	12,800
Seepage from underground workings and open pit	211,000	580
Runoff from open pit	238,000	32,400
Runoff from dirty rock stockpile	278,000	3,800
Outflows		
Evaporation from the pond surface	59,000	negligible
Seepage Losses	Conservatively ignored	negligible
Treatment plant	777,000	5,000

Design parameters for the Holding Pond are summarized below:

- Dead Storage: A volume of 29,000m^3 is assumed for the sediment containment. T he top elevation of the sediments is at 291.5m, 0.5m above the pond bottom.
- Minimum Normal Pond Operating Level: Minimum pond operating level is set at 292.0 m, 1.0 m above the bottom of the pond.
- Normal Operating Volume in the Pond: The normal operation volume of 174,000m^3 will be required to retain contaminated water under average year runoff condition.
- Maximum Normal Operating Level: The maximum water level for an average runoff year is at elevation 293.5m.
- Design Pumping Rate: Water from the Holding Pond will be pumped to the treatment plant at following rates:
  - For an average year: 2,130m^3/d
  - For 1:25 year rainfall event: 5,000m^3/d
- Emergency Spillway Invert Elevation (operation): Determined by the operating and no spill criteria, this elevation was selected at 293.9 m, providing a total storage capacity of 282,000m^3.
- Emergency Spillway Configuration: A trapezoidal spillway channel (3H:1V) was designed with a bottom width of 3 m and a longitudinal slope of 0.5%.
- Dam Crest: Dam crest elevation was designed at 294.5m, provides a 0.3m freeboard above the maximum water level of 294.2m in a 1:100 year hydrologic event during operation.

16.4 Capital Costs

LoM capital costs totaling US$156.1million are summarized in Table 16.4.1. Details supporting this estimate are discussed in this section. Preproduction capital costs are US$86.9million. Ongoing capital accounts for the remaining mine life. Capital cost estimates are in Q1 2008 US constant dollar terms.

Table 16.4.1: LoM Capital Costs (US$000s)

Description	Pre-Production Capital (through Q4-2008)	Ongoing Capital (Q1-2009 to End)	Total Capital
Equipment			
Open Pit Mine	$13,462	$0	**$13,462**
Underground Mine	$7,729	$4,826	**$12,555**
Infrastructure @ Black Fox	$22,989	$1,656	**$24,644**
Black Fox Mill	$0	$0	**$0**
Tailings @ Black Fox	$0	$0	**$0**
Stock Mill	$1,301	$0	**$1,301**
Tailings @Stock	$1,518	$1,432	**$2,950**
Mine Closure	$0	$16,091	**$16,091**
Owner Costs	$25,779	$1,336	**$27,116**
Equipment	**$72,777**	**$25,341**	**$98,118**
Development			
Capitalized Operating Cost	$5,732	$0	**$5,632**
Glacial Till Removal	$4,013	$16,048	**$20,061**
Underground Mined Development	$4,397	$27,774	**$32,171**
Development	**$14,142**	**$43,821**	**$57,963**
TOTAL CAPITAL	**$86,919**	**$69,162**	**$156,081**

Capital cost estimates shown include delivery and erection/assembly charges, where applicable. Tax on equipment is refunded by the government, so a tax provision is not included in the estimate.

16.4.1 Open Pit Mine Capital

Equipment

The estimated cost of mine equipment is shown in Table 16.4.1.1. Mine capital equipment costs were obtained by soliciting price proposals for new equipment, and a fleet of six used Komatsu 105t haul trucks, one used water truck and one used fuel-lube truck. The average frame hours on the haul trucks is 22,000 hours. A third drill is included as a lease cost starting in October 2010 for a 1.5 year time period. A lease cost of US$15,000 per month has been included for the life of the mine for a contractor explosive truck.

A contingency of 10% is included in the estimate.

Table 16.4.1.1: Open Pit Mine Capital Costs (US$000s)

Equipment	Size	Model	Age	Units	Unit Cost (US$000s)	Total (US$000s)
Open Pit Mine						
Rotary Drill	165mm	DM45	New	1	$1,025	$1,025
Hydraulic Drill	110mm to165mm	CM785	New	1	$751	$751
Mass Excavator	$4m^3$	Cat 365CL	New	1	$661	$661
Mass Excavator	$10m^3$	O&K RH 90	New	1	$2,950	$2,950
Front End Loader	$6.5m^3$	Cat 988HL	New	1	$755	$755
Truck	100t	Komatsu 785-3	Used 1997	5	$295	$1,475
Truck	100t	Komatsu 785-3	Used 1997	1	$212	$212
Bucket (spare)	$10m^3$	O&K	New	1	$273	$273
Dozer - Track	310hp	Cat D8T	New	1	$591	$591
Dozer - Track	410hp	Cat D9T	New	1	$890	$890
Grader	16ft	Cat 16H	New	1	$772	$772
Water Truck	5k-gal	Cat 725	Used	1	$260	$260
Fuel Lube Truck	1k-gal	L8000	Used	1	$44	$44
Mechanic Truck	-	Sterling	New	1	$250	$250
Welding/Crane Truck	-	Sterling	New	1	$250	$250
Pickup Truck	3/4t		New	15	$30	$450
Light Plant	-	Magnum MTL3060	New	4	$8	$32
Sanding/Stemming Truck	-		New	1	$80	$80
Forklift for Warehouse	-	Cat TL1255	New	1	$149	$149
Backhoe Loader	97hp	Cat 430E Tool Carrier	New	1	$128	$128
Flatbed Truck	-		New	1	$40	$40
Crew Vans	-		New	1	$35	$35
Skid Loader	-	Cat 272C	New	1	$54	$54
Forklift for Mill		Load Lifter 2414	New	1	$104	$104
ATV	-		New	1	$6	$6
Open Pit Equipment						**$12,238**
Contingency @ 10%						$1,224
Mine Capital						**$13,462**

Glacial Till Stripping

Surface related capital costs also include a provision for the removal of glacial till material prior to the drilling and blasting of overburden. This work will be performed by a contractor in two phases. Phase 1 work will commence Q3 2008 and will end Q1 2009. Phase 2 will commence Q2 2010 and will end Q1 2011. Costs are shown in Table 16.4.1.2.

Table 16.4.1.2: Glacial Till Removal Costs (US$000s)

Description	Material (kt)	Unit Cost US$/t	Total (US$000s)
Phase I Pit	3,585	$1.76	$6,309
Phase II Pit	7,220	$1.90	$13,752
Total	**10,822**	**$1.85**	**$20,061**

16.4.2 Underground Mine Capital

Equipment

The estimated cost of underground capital, totaling $11.6million is shown in Table 16.4.2.1. Mine equipment costs were obtained by soliciting price proposals for all new equipment. A contingency of 15% is included in the estimate.

The underground development will commence Q3 2008 to allow sufficient lead-time to develop the production stopes to supply a continuous source of ore for the mills.

Table 16.4.2.1: Underground Capital Costs (US$000s)

Description	Model/Type	No. of Units	Unit Cost (US$000s)	Total Cost (US$000s)
Mobile Equipment				
2-Boom Jumbo	Tamrock H205D	1	$485	$485
2-Boom Jumbo	MTI DR2SB	2	$858	$1,716
1-Boom Jumbo	Tamrock H105M	1	$315	$315
Scooptram	Wagner ST3.5	2	$345	$690
LHD	MTI LT650	2	$479	$956
LHD Remote Cont.	MTI LT650 Accessory	1	$75	$75
Backfill Mucker	-	1	$175	$175
Truck	Tamrock EJC430 28t	2	$415	$830
Truck	MTI DT3004 28t	1	$625	$625
Scissor Lift	Getman A64	2	$225	$450
Scissor Lift	UVT-SL6	1	$339	$339
ANFO Loader	Getman	2	$295	$590
Fuel/Lube Truck	-	1	$396	$396
Boom Truck	-	1	$296	$296
Man Carrier	MTI MUT210	2	$191	$382
Tractor	Ford	3	$50	$150
Equipment				**$8,619**
Fixed Equipment				
Cement Silo	-	1	$155	$155
Ventilation Fan	250hp for E. Vent. Raise	1	$50	$50
Ventilation Fan	100hp for W. Vent. Raise	1	$100	$100
UG Fans	for Internal Raises	2	$75	$150
Water Mgmt. System	Upgrade	1	$150	$1,000
Underground Shop	No. 1	1	$100	$100
Underground Shop	No. 2	1	$250	$250
Switchgear &Feeder	for E. Ventilation Raise	1	$394	$394
Air Compressor	-	1	$100	$100
UG Sump- (incl in wtr mgmt)	Pumps and piping	1	$56	$0
Fixed				**$2,299**
subtotal				*$10,917*
Contingency @15%				$1638
Underground Capital				**$12,555**

Underground Development

Underground mine development costs are shown in Table 16.4.2.2. Development cost estimates presented in the table do not include labor. Labor allocation for development was estimated separately and is shown as a separate line item.

Table 16.4.2.2: Underground Development Costs (US$000s)

Description	Unit Costs (US$/m)	Initial (m)	Ongoing (m)	Total (m)	Initial (US$000s)	Ongoing (US$000s)	Total (US$000s)
Development							
Ramp (5m x 5m)	$854.53	1,872	0	**1,872**	$1,492	$0	**$1,492**
Ramp (4m x 5m)	$678.53	354	585	**939**	$221	$365	**$586**
Ramp (4m x 4m)	$608.97	4,028	4,503	**8,531**	$2,299	$2,570	**$4,869**
Vent. Raise (4m x 4m)	$6,268.84	476	0	**476**	$2,984	$0	**$2,984**
Vent Raise (3m x 3m)	$2,090.59	444	576	**1,020**	$928	1,204	**$2,132**
subtotal	**-**	**7,174**	**5,664**	**12,838**	**$7,925**	**$4,139**	**$12,064**
Stope							
Stope Ramp – Cut 1	$578.56	1,953	5,789	**7,742**	$1,051	3,115	**$4,166**
Stope Ramp – Cuts 2-6	$409.21	2,328	13,012	**15,340**	$953	5,324	**$6,277**
subtotal	**-**	**4,281**	**18,801**	**23,082**	**$2,003**	**$8,439**	**$10,443**
Labor	-	-	-	-	$2,477	$7,186	**$9,663**
Total Development	**-**	**-**	**-**	**-**	**$12,405**	**19,766**	**$32,171**

16.4.3 Black Fox Mine Infrastructure Capital

Infrastructure costs associated with Black Fox facilities are shown in Table 16.4.3.1. Major items include US$10million for the truck shop and site development. Capital costs were developed by SE and include the Black Fox Mill along with the estimate of infrastructure. The infrastructure cost presented here was excerpted from the SE estimate by SRK. Other infrastructure costs are minimal as the mine is located in a well-established mining district and is very close to roads and power interconnections.

Existing power and substation are sufficient to operate the mine and ancillary systems. There is a complete laboratory at the Stock Mill facility, so one is not required at Black Fox.

Table 16.4.3.1: Black Fox Infrastructure Costs (US$000s)

Description	Total (US$000s)
Direct Costs	
Fresh Water & Utilities	$2,074
Substation & Power Distribution	$0
Site Development (incl. bridge)	$3,223
Truck Shop & Dry	$6,139
Administration Building	$1,648
Laboratory	$0
Fuel Depot and Ready Line	$273
subtotal	**$13,358**
Indirect Costs	
Freight	$403
Contractor Indirects	$2,740
Construction Equipment	$1,146
Spare Parts	$26
Initial Fills	$0
Vendor Representatives	$38
QA/QC	$108
Survey Verification	$37
EPCM Services	$2,875
Commissioning Support	$223
subtotal	**$7,595**
Contingency @ 17.6%	$3,692
Total Infrastructure	**$24,645**

16.4.4 Black Fox Mill & Tailings Dam Capital

A capital cost estimate for the construction of a mill and tailings facility at the Black Fox Project site was prepared by SE and AMEC, respectively. These estimates are shown in the following tables.

However, subsequent studies following the recent purchase of the Stock Mill has resulted in Apollo's decision to postpone the construction of the Black Fox Mill and Tailings Dam. For this reason, the capital costs for these facilities presented in this section, are not included in the project economics.

Table 16.4.4.1: Black Fox Mill Costs (US$000s)

Description		Total (US$000s)
Direct Costs		
Crushing		$7,344
Grinding		$10,827
Pre-Leaching		$3,324
Leaching & CIP		$5,725
Carbon Stripping & Regeneration		$2,288
Tailings/Water Treatment		$1,899
Refining		$1,073
Reagents		$404
	subtotal	**$32,884**
Indirect Costs		
Freight		$1,569
Contractor Indirects		$6,744
Construction Equipment		$1,861
Spare Parts		$522
Initial Fills		$291
Vendor Representatives		$592
QA/QC		$266
Survey Verification		$90
EPCM Services		$7,080
Commissioning Support		$500
	subtotal	**$19,535**
Contingency @ 17.6%		$9,236
	Total Black Fox Mill	**$61,655**

Table 16.4.4.2: Black Fox Tailings Dam Costs (US$000s)

Description	Total (US$000s)
East Tailings Dam Costs	
Mob/Demob	$300
Basin Clearing	$235
Dewatering	$100
Excavation – Cut-Off Trench	$69
Grubbing & Stripping	$1,170
Zone 1 – Silt Sand Fill	$6,821
Zone 2 – Filter/Transition	$3,629
Zone 3 – Rock Fill	$1,565
Zone 4 – Dirty Waste Rock	$2,016
Zone 5 – Rock Fill	$1,378
Zone 6 – Rip Rap	$701
Zone 7 – Silty Clay	$59
Geotextile	$177
Wick Drains	$1,919
Emergency Spillway	$20
Pump Station	$500
Pipeline-Reclaim & Treatment	$920
Spill Collection Pond	$75
Power line	$360
Access Road	$240
Instrumentation	$20
subtotal	**$22,850**
Contingency @ 17.6%	240
Total Black Fox Mill	**$26,277**

16.4.5 Stock Mill & Tailings Dam Capital

Process capital costs shown in Table 16.4.5.1 and include all costs to startup the Stock Mill. The Stock Mill will be purchased by Apollo in April 2008 and has been under care and maintenance during recent years. An assessment of the facility has concluded that minimal effort would be required to begin operations. However, AMEC has recommended that some additional testing of the tailings dam is required. The model therefore has included a US$1million provision for this testwork and possible improvements, which may be required. A detailed discussion of the process design and equipment requirements can be found in Section 14 of this report.

Table 16.4.5.1: Stock Mill Capital Costs (US$000s)

Description	Total (US$000s)
Mill Refurbishment	
Mill Refurbishment Labor	$118
Power	33
Reagents	22
Grinding Steel	14
Maintenance Supplies	7
Miscellaneous Operating Supplies	1
First Fills	349
Steel Liners for Primary Mill	65
Spare Parts	522
Subtotal	***$1,131***
Tailings Dam Upgrades	
Provision for Dam Upgrade	$1,000
Phase 4 Raise with Toe Dam	$320
Phase 5 Raise with Toe Dam	$1,246
Subtotal	*$2,566*
Contingency @ 15%	$554
Total Stock Mill Refurbishment	**$4,251**

16.4.6 Closure Costs

Closure costs for Black Fox are shown in Table 16.4.6.1. These costs are estimated to be US$16.1million and are comprised as follows:

Table 16.4.6.1: Black Fox Closure Costs (US$000s)

Description	Total (US$000s)
Closure Costs	
Secure Openings – Raises	$50
Secure Openings – Portal/Decline	$9
Secure Openings – Open Pit	$150
Stabilize Subsurface Workings	$20
R&D Subsurface Buildings/Infrastructure	$432
Transportation Corridors & Roads	$15
Concrete Structures	$30
Petroleum Chemicals Explosives	$5
Test & dispose of HC contaminated soils	$12
Tailings Facility	$13,454
Waste Rock Overburden Stockpile	$12
Stabilize Impoundment Structures	$23
Restore Site Drainage	$12
Monitoring Program	$441
Subtotal	*$14,665*
Indirect Costs	
Mobilization & Demobilization	$137
Bond	$55
Engineering, Contracts, Supervision	$411
Contingency	$823
Taxes	$0
Subtotal	*$1,426*
Total Closure Cost	**$16,091**

16.4.7 Owner Costs

Owner capital costs total US$27.1million as is shown in Table 16.4.7.1. Owner costs include US$20million for the purchase of the Stock Mill as well as ponds and ditches, which are required since the on-site mill construction was postponed. In addition, provisions for on-going detailed engineering and design work, permitting, start up and commissioning and other corporate overheads are included.

Table 16.4.7.1: Owners Costs (US$000s)

Description	Total (US$000s)
Owner Costs	
Purchase Stock Mill	$20,000
Black Fox – Holding Pond	$3,067
Black Fox – Overburden Stockpile Collection Ditches	$444
Black Fox – Dirty Waste Pile Run-off Ditches	$244
Detailed Engineering	$600
Ongoing Permitting	$1,738
Start-up & Commissioning	$250
Contract Crusher Mob/Demobilization	$35
Corporate Overheads	$300
Subtotal	*$26,678*
Contingency @15%	$438
Total Closure Cost	**$27,116**

16.5 Operating Costs

LoM operating costs are summarized in Table 16.5.1. Details supporting this estimate are discussed in this section. Operating cost estimates are in Q1 2008 US constant dollar terms.

Table 16.5.1: Operating Cost Summary

Description	LoM Total (US$000s)	Unit Cost (US$/total-t)	Unit Cost (US$/ore-t)*
Open Pit Mining	$107,348	$2.13/t	$24.68/t
Underground Mining	$117,083	-	55.39/t
Mine G&A	$42,576	-	$0.81/t
Holt Toll Mill	$120,256	-	$39.54/t
Stock Mill	$84,265	-	$24.63/t
Black Fox Mill	$0	-	$0.00/t
G&A	$19,907	-	$3.08/t
Total	**$485,704**	**-**	**$75.15/t** **$385.29/Au-oz**

*Weighted average over the LoM.

16.5.1 Labor Costs

Labor costs are based upon a defined work force. Salaries and hourly rates applied are in accord with current values in the region. All salaried staff and hourly paid tradesmen will work a 40-hour week. Operating personnel will provide continuous coverage.

Pay scales are based on actual rates/salaries in the Timmins area, are shown in Table 16.5.1.1, and are as of Q1 2008. A payroll burden of 30% is applied to all salaries and wages. In addition, a 35% bonus allowance is applied to hourly personnel.

Personnel will reside in the neighboring communities and will be responsible for transportation from their domicile to the place of work. Apollo will not provide transportation, on a normal basis.

Table 16.5.1.1: Labor Rates (Q1 2008)

Description	Rate	Bonus	Burden
Miner 1	$26.35	$35.57	$46.24
Miner 2	$25.91	$34.98	$45.47
Miner 3	$25.69	$34.68	$45.09
Miner 4	$25.00	$33.75	$43.88
Hoistman	$26.73	$36.09	$46.91
Hoistman Temp	$26.35	$35.57	$46.24
Shaftman	$27.63	$37.30	$48.49
Cage/Skiptender	$24.59	$33.20	$43.16
Serviceman	$24.59	$33.20	$43.16
Labour / Geo Tech	$23.93	$32.31	$42.00
Labour / Security	$20.00	$27.00	$35.10
Mill Lead Hand	$19.08	$25.75	$33.48
Mill Op 1	$17.84	$24.08	$31.30
Mill Op 2	$16.95	$22.89	$29.75
Mill Op 3	$16.07	$21.70	$28.21
Mill Op 4	$14.57	$19.67	$25.58
Mill Op 5	$12.81	$17.29	$22.48
Instrument Tech Sr.	$30.00	$40.50	$52.65
Instrument Tech	$28.00	$37.80	$49.14
Lead Hand	$29.32	$39.58	$51.46
Licensed Trades	$29.00	$39.15	$50.90
Unlicensed Trades	$28.00	$37.80	$49.14
Apprentice 4th Year	$19.36	$26.13	$33.97
Apprentice 3rd Year	$17.56	$23.70	$30.81
Apprentice 2nd Year	$15.76	$21.27	$27.65
Apprentice 1st Year	$13.96	$18.84	$24.49
Mechanic	$29.55	$39.89	$51.86
Heavy Equip Op	$29.55	$39.89	$51.86
Loader Op	$28.00	$37.80	$49.14
Tailing Loader Op	$23.00	$31.05	$40.37
Student Rate	$18.65	$25.18	$32.73

16.5.2 Open Pit Operating Costs

A summary of the estimated mine operating cost are shown in Table 16.5.2.1. Mine operating costs were updated with the current fuel and tire prices and Montana Tunnels actual operating costs. LoM open pit mine operating costs will total US$107million, or US$2.13/total-ton of material moved.

Table 16.5.2.1: Open Pit Mine Operating Cost (US$000s)

Description		LoM Total (US$000s)
Drilling		
Salaries & Wages		$8,254
Lease for CM785 Drill		$306
Fuel & Lube		$4,345
Consumables, Parts		$2,693
	Drilling	**$15,598**
		$0.309/tot-t
Blasting		
Salaries & Wages		$2,449
Lease for Powder Truck		$1,575
Consumables, Parts		$11,943
	Blasting	**$15,967**
		$0.317/tot-t
Loading		
Salaries & Wages		$7,971
Fuel & Lube		$4,223
Consumables, Parts		$3,657
	Loading	**$15,851**
		$0.314/tot-t
Hauling		
Salaries & Wages		$14,422
Fuel & Lube		$9,593
Consumables, Parts		$5,593
	Hauling	**$29,608**
		$0.587/tot-t
Support		
Salaries & Wages		$10,790
Fuel & Lube		$13,029
Consumables, Parts		$6,506
	Support	**$30,324**
		$0.602/tot-t
	Open Pit Mining	**$107,348**
		$2.13/tot-t

16.5.3 Underground Operating Costs

Underground mining operating costs were determined using first principle engineering, known consumption rates from previous mining experience at the project, and local vendor supply bids. Stoping costs are shown in the Table 16.5.3.1.

Underground development costs are discussed in Section 16.4.2.

Table 16.5.3.1: Stope Operating Costs

Description	LoM Total (US$000s)
Ore Production	
Production Labor	$35,239
Maintenance Labor	$15,750
Stoping	$22,858
Ore Haulage to Surface	$4,532
Backfill (3% Cement)	$19,059
Rockfill	$18,096
Mine Dewatering	$1,550
Underground Mining	**$117,083**
	$55.39/ore-t

16.5.4 Mine G&A Operating Costs

Mine G&A costs are shown in Table 16.5.4.1. Mine G&A are includes all overhead costs shared by open pit and underground mining operations.

Table 16.5.4.1: Mine G&A Costs (US$000)

Description	LoM Total (US$000s)
Open Pit G&A	
Salaried Management	$10,884
O&M Costs	$1,701
OP G&A	**$12,858**
	$0.25/op-t
Underground G&A	
Salaried Management	$6,095
O&M Costs	$8,386
UG G&A	**$14,479**
	$6.85/ug-t
Technical Services	
Technical Services (Salaries)	$7,244
Geology & Ore Control	$8,268
Tech Services	**$15,512**
	$0.30/ore-t
Open Pit Mining	**$42,576**
	$0.81/ore-t

16.5.5 Processing Costs

The project considered three milling options over the course of the Feasibility Study.

- Toll mill at St. Andrews' Holt Mill,
- Recently purchased Stock Mill, and
- Build a new on-site mill at Black Fox.

The feasibility concluded that the project will operate the Stock Mill and toll mill at the Holt facility. Therefore, operating costs for the Black Fox Mill, although shown in this section, are not included in the project economics.

Holt Toll Mill

Toll mining and associated costs are shown in Table 16.5.5.1. On-site crushing of toll ore is not required as the crusher at the Holt Mill is capable of processing Black Fox ores. However, included is a provision for an Apollo loader operator for RoM ore handling.

Transportation to the Holt mill includes contract loading and hauling of ore to the Holt Mill. The estimate includes a fuel surcharge. Processing costs of US$30/t are per the contract with St. Andrews, and is all-inclusive.

Table 16.5.5.1: Holt Toll Mill Costs (US$000s)

Description	LoM Total (US$000s)
On-Site Crushing (Contractor)	
Salaries & Wages	$2,146
On-site Crushing and Loading at Black Fox	$0
On-Site Crushing	**$2,146**
	$0.71/t
Transportation to Mill (Contractor)	
Labor	$0
Loading & Hauling (includes labor)	$26,866
Transportation to Mill	**$26,866**
	$8.83/t
Processing	
Contract Milling Rate	$91,244
-	$0
Tech Services	**$91,244**
	$30/t
Holt Toll Mill	**$120,256**
	$39.54/ore-t

Stock Mill

The Stock Mill has a nominal capacity of 1,100tpd. On-site crushing is required to supplement the crushing circuit at the Stock Mill. Therefore, open pit and underground ore mined at Black Fox will be crushed on-site and loaded to OTR trucks by a contract crusher. Contracted OTR trucks will transport the ore to the Stock Mill. LoM costs are shown in Table 16.5.5.2.

LoM costs are estimated to be US$84.3million or US$24.63/t.

Table 16.5.5.2: Stock Mill Operating Costs (US$000s)

Description	LoM Total (US$000s)
Labor	
Salaried	$7,439
Hourly	$11,026
Labor	**$18,465**
	$5.40/t
On-Site Crushing (Contractor)	
Salaries & Wages	$2,208
Crushing & Loading	$7,631
On-Site Crushing	**$9,839**
	$2.88/t
Transportation to Mill (Contractor)	
Labor	$0
Transportation Only	$22,517
Transportation to Mill	**$22,517**
	$6.58/t
Consumables	
Reagents	$9,487
Grinding Steel	$6,045
Spare Parts	$2,941
Consumables	**$18,473**
	$5.40/t
Power	
Power	$14,972
-	$0
Power	**$14,972**
	$4.38/t
Stock Mill	**$84,265**
	$24.63/t

Black Fox Mill

If constructed, the Black Fox Mill will have a nominal capacity of 1,500tpd. On-site crushing or off-site transportation will not be required. Potential LoM costs are shown in Table 16.5.5.3.

As with the Stock Mill a provision for an Apollo loader operator for RoM ore handling is included. LoM costs are estimated to be US$48.0million or US$13.84/t.

Table 16.5.5.3: Black Fox Mill Operating Costs (US$000s)

Description		LoM Total (US$000s)
Labor		
Salaried		$5,739
Hourly		$8,506
	Labor	**$14,245**
		$4.11/t
On-Site Crushing (Contractor)		
Salaries & Wages		$3,355
Crushing & Loading		$0
	On-Site Crushing	**$3,355**
		$0.97/t
Consumables		
Reagents		$8,292
Grinding Steel		$4,676
Spare Parts		$2,268
	Consumables	**$15,236**
		$4.53/t
Power		
Power		$15,177
-		$0
	Power	**$15,177**
		$4.38/t
	Stock Mill	**$48,014**
		$13.84/t

16.5.6 G&A Costs

LoM G&A costs have been estimated based upon similar operations and are shown in Table 16.5.6.1. LoM G&A costs will be US$19.9million, or, US$3.08/t.

Table 16.5.6.1: G&A Costs

Description		LoM Total (US$000s)
Labor		
Management Support (Salaried & Hourly)		8057
-		-
	Labor	**$8,057**
		$1.25/t
Environmental Management		
Water Quality Monitoring		$1,568
Geotechnical Investigations		$175
CEAA Compensation		$160
Other Costs		$1,400
	Environmental	**$3,303**
		$0.51/t
Operating Supplies		
O&M Supplies		$8,547
-		$0
	Supplies	**$8,547**
		$1.32/t
	G&A Cost	**$19,907**
		$3.08/t

16.6 Markets

Markets for doré are readily available. Gold markets are mature, global markets with reputable smelters and refiners located throughout the world. Demand is presently high with prices for gold showing remarkable increases during recent times. The 36-month average London PM gold price fix through March 2008 is US$623/oz.

16.7 Contracts

The Black fox Project has signed 5 contracts which will be directly associated with operations. These contracts, shown below, are also modeled in the economic analysis.

- Glacial Till Removal. Leo Alarie and Sons Limited entered into an agreement with Apollo to remove glacial till in preparation to overburden removal in the open pit operation. This work will be completed in 2 phases. Phase I work will occur from Q3 2008 to Q1 2009 for US$1.76/t and Phase II work will be from Q2 2010 to Q2 2011 at a cost of US$1.90/t.

- On-Site Crushing. Leo Alarie and Sons Limited entered into an agreement with Apollo to crush and load ore for the Stock Mill. On-Site crushing and loading is quoted at US$2.3/t.

- Truck Transportation to Stock and Holt Mills. North-West Transport Inc. entered into an agreement to transport ore to the Stock Mill and to load and haul ore to the Holt Mill. Hauling to Stock is quoted at $5.25/t plus a fuel surcharge. Load and hauling to Holt is quoted at $1.00/t for loading and $6.25/t hauling plus a fuel surcharge.

- Toll milling at St. Andrew's Holt Mill. Apollo entered into an agreement for the tolling of ore at St. Andrews' Holt Mill. The contract for US$30/t is all inclusive and is for a nominal 1,400tpd. However additional material could be processed as the contract allows for this and capacity exists at the mill.

- Refining of Doré. Smelting and refining of doré will be performed under the terms of an agreement with Johnson Matthey. Refining will be at a cost of US$0.55 per total ounce received. Payable gold will be 99.62%. Transportation and insurance is also included in the contract. Given a weekly shipment schedule, transportation and insurance is assessed at US$850/shipment up to a value of US$400,000 and US$0.70 for every $1000 above US$400,000.

Currently, there are no hedging, forward selling or other agreements/contracts related to Black Fox.

16.8 Environmental Considerations

16.8.1 Regulatory Considerations

"A number of environmental issues will be considered for the permitting and approvals process related to the Black Fox Project. These environmental issues include: the removal of all existing site infrastructure and the construction of new infrastructure, effects to terrestrial habitat, potential effects to fisheries habitat, groundwater and area surface waters, as well as air quality and related noise issues associated with open pit mining activities.

As the Project continues to move forward, further contact with various government agencies will be established and a formal public and stakeholder consultation process initiated. There are a number of permitting considerations that must be addressed in order for the project to advance. These considerations include, but are not limited to, the following:

- Amendments to existing and/or application for new Permits to Take Water (MOE);
- Amendment to existing C of A for Air (MOE);
- Application for a new C of A for Industrial Sewage Works (MOE);
- Potential Fisheries Act authorization (i.e., Letter of Advice) (Department of Fisheries and Oceans);
- Filing of Mine Closure Plan with financial assurance (MNDM); and
- Authorizations/easements for site access requirements (MTO).

It is planned to commence with applicable provincial permitting and approval application submissions as early as mid May 2007, with receipt of approvals anticipated for the end of 2007 or early 2008. Site development would commence shortly thereafter.

16.8.2 Mine Development Considerations

Since Apollo purchased a 100% interest in the Glimmer Mine in 2002, exploration initiatives have supported the preparation of a positive feasibility study for open pit development, which is to be supplemented with an underground mine operation. For this mining scenario, Apollo has undertaken various environmental baseline, geochemical, geotechnical, hydrogeologic, metallurgical and other studies in support of the Black Fox Project. The Black Fox Project will encompass a number of major modifications to the site due to the development of an open pit mining operation and proposed underground mine. The major aspects associated with the proposed Project permitting would include the following:

- Removal of existing site infrastructure and the construction of new infrastructure to facilitate the open pit development;
- Effects to terrestrial habitat, due to the development of new overburden and waste rock piles;
- Potential effects to fisheries habitat, due to the placement of overburden and/or waste rock stockpiles adjacent to, or near, local watercourses;
- Potential effects to groundwater flow paths, associated with a potentially increased rate of dewatering of mine workings;
- Potential effects to area surface waters (particularly with respect to Froome Lake), due to a potentially increased mine dewatering rate;
- Potential effects to area surface waters resulting from site drainage, and treated water discharge from the water treatment plant;
- Air quality and related noise issues associated with open pit mining activities;
- A possible crossing of Highway 101 to stockpile organic material north of the highway; and
- Public and stakeholder consultation.

Activities associated with the development of the proposed open pit and underground mine, which will require the removal of existing infrastructure and the construction of new infrastructure, will be subject to permit authorizations from various regulatory agencies. In the event that the proposed mine development would result in significant effects on fish habitat, a federal authorization under the Fisheries Act would be required and the Project could be subject to a comprehensive level environmental assessment under the Canadian Environmental Assessment Act. However, for the proposed open pit and underground mine scenario, and the proposed configurations of the waste rock and overburden stockpiles, potential impacts on local fish habitat have been mitigated and/or eliminated. The only other potential for federal involvement could be related to the construction of any water intake and/or discharge structures, which could require a Letter of Advice from the Department of Fisheries and Oceans.

In developing the small open pit and further advancing underground development, surface runoff and groundwater inflows into the mine are expected to increase. Hydrogeological and hydrological investigation programs have been undertaken to characterize the effects of the potential increase of such inflows into the proposed mine development on the surrounding environment, including Froome Lake. These investigations indicate that the potential for significant effects, as a result of the additional inflows and/or diversions, can be mitigated to reduce such effects.

Based on a review of Ministry of Environment publications relating to sound level limits, it is assumed that the area surrounding the Black Fox Project would be considered as a Class 3 Area (Rural). A comprehensive noise assessment of the site will be completed, as required, in support of an application for a C of A (Air) for all mine related equipment, such as ventilation exhaust fans, emergency generators and maintenance areas, as well as emissions from the mill operation. Air emissions are anticipated to be minimal at the site and will be primarily limited to fugitive particulate emissions from material handling equipment (loaders, trucks, conveyors/crushers) and products of combustion from diesel equipment and propane heaters.

Special requirements for pit blasting may be required due to the proximity of the pit to Highway 101 and will need to be discussed with the Ontario Ministry of Transportation (MTO). Some settlement of the highway could also occur as a result of the drawdown of the water table. The potential for settlement is to be evaluated during the feasibility level design stage. Public and stakeholder consultation is required for all new and existing proposed mine expansion projects. The public consultation program will entail an open house for the general public, continued consultation with the First Nations, and possibly meetings with municipal representatives from Matheson and/or local residents, recreational groups and small business owners.

The First Nation (FN) community having an interest in this mine development project is the Wahgoshig First Nation. The community of approximately 250 registered members is located 25 km northeast of the mine site. A series of meetings were held by Apollo with the Wahgoshig First Nation throughout 2006, which resulted in the signing of a Memorandum of Understanding (MOU) in January 2007 between both parties. The key aspects of the MOU include provisions for training and ongoing communication. The MOU also outlines an agenda and process for negotiating an Impact Benefit Agreement (IBA), which will include such topics as employment, *training, business opportunities and financial compensation when the Black Fox Project moves from an exploration phase to a production phase.*

16.8.3 Mine Reclamation and Site Closure

Upon the cessation of mining activities, Apollo will reclaim the Black Fox Project site as required under Section VII of the Mining Act. Opportunities to progressively reclaim the site will be exploited and progressive rehabilitation efforts maximized over the life of the operation where possible. After reclamation and closure, there will be a number of facilities requiring monitoring and maintenance. Remaining site facilities will include: an open pit; capped mine shaft, raise(s) and portal; as well as reclaimed tailings and waste rock and overburden stockpiles.

At the conclusion of mining, all openings to surface will be sealed with reinforced concrete caps designed in accordance with Schedule 1 of Ontario Reg. 240/00, the Mine Rehabilitation Code of Ontario. Crown pillars will be assessed for long term stability and rehabilitation measures implemented accordingly, with the objective of ensuring adequate site safety.

Restoration of the open pit will involve flooding (i.e., creation of a pit lake) in conjunction with backfilling of the pit with waste rock (to prevent sulphide oxidation). The shallow groundwater regime in the area, along with benign country rock, strongly suggests that flooding of the pit is a practical closure option. It is expected that a one-time addition of lime to the pit water will be required to neutralize any acidity of (partly oxidized) waste rock.

All non-closure related infrastructure will be removed from the site. All site buildings will be demolished and removed to an off-site licensed landfill facility as required. All building foundations will be demolished, and subsequently covered with overburden and seeded with an appropriate vegetative mixture. All non-essential site distribution services, including electrical power, water, tailings, sewage and gas lines, will be removed from site. Below ground services will be decommissioned and left buried. All mobile and fixed equipment will be removed from site. Any remaining inventories of chemicals or petroleum products will be returned to the appropriate vendors and hazardous wastes will be disposed of using appropriately licensed waste haulers and contractors.

The tailings facility, and "clean" waste rock and overburden stockpiles, will be designed and constructed to ensure long-term physical stability. Revegetation of all "clean" waste rock and overburden piles will be carried out with the objective of creating a self-sustaining vegetative cover. It is planned to move the "dirty" waste rock into the open pit for final disposal, where it will be submerged to minimize oxidation and metal leaching. Closure of the tailings impoundment will consist of either a soil cover or a water cover. Ongoing assessment programs will be implemented as part of mine operations to continually monitor chemical stability.

All site related drainage channels or water management structures created as a result of mining operations or closure initiatives will be removed or stabilized. Design criteria for the design of remaining water structures such as drainage ways or spillways would be developed in accordance with current engineering standards. Where appropriate, suitable erosion protection will be designed and installed.

It is anticipated that all reclamation activities would be completed within a period of five years upon the cessation of site-related mining activity. Monitoring would be implemented to monitor the effectiveness of closure measures.

The treatment of excess tailings pond water would continue until the cyanide and arsenic levels become sufficiently low for discharge of site waters directly to the environment."

16.9 Taxes and Royalties

Income tax was not considered in this report. Provincial and government sales taxes are refundable for mining operations and therefore are not included in this analysis. Import duties and other fees associated with capital items are included in the capital cost estimate.

There are no royalty obligations associated with the current Black Fox reserves and resources.

16.10 Economic Analysis

The technical-economic results summarized in this section are based upon work performed by Apollo's engineers and consultants and has been prepared on an annual basis. The economic model was developed by SRK, and is shown in Exhibit 16.1. All costs are in Q1 2008 US constant dollars.

16.10.1 Model Inputs

The economic model, presented in Exhibit 16.1, is pre-tax and assumes 100% equity to provide a clear picture of the technical merits of the project. Assumptions used are discussed in detail throughout this report and are summarized in Table 16.10.1.1

Table 16.10.1.1: Technical Economic Model Parameters

Model Parameter	Technical Input
General Assumptions	
Pre-Production Period	15 months
Mine Life	8.75 years
Operating Days per year	360 days/yr
Production Rate (avg.)	2,500tpd
Market	
Discount Rate	5%
Gold Price	US$750.00/oz
Royalty	
Private Royalty	none

A 15-month pre-production rate is assumed to allow for pre-stripping and mine development. The mine will have an estimated life of 8.75 years given the reserves described in this report and the assumed 2,500tpd production rate.

Revenue from gold sales are based upon a market price of US$750/oz. Gold treatment and refining charges are at US$0.55/oz plus a charge transportation and insurance of approximately US$0.70 per US$1,000 of shipment value. Refining, transportation and insurance costs are charged against gross revenues.

16.10.2 LoM Plan and Economics

The SRK LoM plan and economics are based on the following:

- A gold price of US$750/oz;
- Probable reserves, no resources are included;
- A mine life of 8.75 years, at a designed rate of 875ktpy;
- An overall average metallurgical recovery rate of 95% Au, over the LoM;
- A cash operating cost of US$75.40/t-milled, US$386.57/oz-Au;

- Initial capital costs of US$86.9million. LoM capital costs are estimated to be US$156.1million being comprised of US$58.0million for capitalized development and US$98.1million for mine equipment;
- Mine closure cost is US$16.1million; and
- No salvage value is modeled.

The base case economic analysis results, shown in Table 16.10.2.1, indicate a pre-tax net present value of US$227.1million at a 5% discount rate with an IRR of 62%.

Table 16.10.2.1: Technical Economic Results ($000s)

Description		Technical Input or Result
Ore		
Open Pit		
Waste		56,881kt
Ore		4,350kt
Total		**61,231kt**
s/r		13.0
Grade		5.218gpt-Au
Contained Gold		729koz
Underground		
Total Development		35,920m
Ore		2,114kt
Grade		8.82gpt-Au
Contained Gold		599koz
Mill		
Ore Treated		
Holt Toll Mill		3,116kt
Stock Mill		3,348kt
Black Fox Mill		0kt
Total		**6,464kt**
Ore Grade		
Holt Toll Mill		6.30gpt-Au
Stock Mill		6.47gpt-Au
Black Fox Mill		0.00gpt-Au
Total		**6.39gpt-Au**
Contained Gold		
Holt Toll Mill		628koz
Stock Mill		700koz
Black Fox Mill		0koz
Total		**1,328koz**
Recovered Gold		
Holt Toll Mill		603koz
Stock Mill		665koz
Black Fox Mill		0koz
Total		**1,268koz**
Revenue ($000s)		
Gross Revenue		**$945,455**
Refining & Transportation Charges		$1,615
Net Smelter Return		**$943,840**
Royalty		$0
Gross Income From Mining		**$943,840**
Realized Price (Gold)	*US$748.72/oz-Au*	
Operating Cost ($000s)		
Open Pit Mine		($107,348)
Underground Mine		($117,083)
Mine G&A		($42,576)
Holt Toll Mill		($120,256)
Stock Mill		($84,265)
Black Fox Mill		$0
G&A		($19,907)
Operating Costs		**($485,704)**
	US$385.29/oz-Au	
	US$75.15/t-milled	
Cash Operating Margin		**$458,136**
	US$363.43/oz-Au	
	US$70.88/t-milled	
Capital Cost		
Equipment		($98,118)
Development (Capitalized)		($57,963)
Total Capital		**($156,081)**
Cash Flow		**$302,055**
($NPV5_{\%}$)		**$227,081**
IRR		**62%**

16.10.3 Sensitivity

Sensitivity analysis for key economic parameters are shown in Table 16.10.3.1. This analysis suggests that the project is most sensitive to market price. Operating costs are slightly more sensitive than capital costs due to the many operating functions associated with the project. Also, the purchase of the existing Stock Mill resulted in a lower than 'typical' capital cost for this project, which has the effect of making capital costs less sensitive.

Table 16.10.3.1: Project Sensitivity ($NPV_{5\%}$, US$000's)

Description	-10%	-5%	Base Case	+5%	+10%
Gold Price	$152,715	$189,898	**$227,081**	$264,264	$301,447
Operating Costs	$266,356	$246,719	**$227,081**	$207,443	$187,806
Capital Costs	$235,932	$231,236	**$227,081**	$222,926	$218,770

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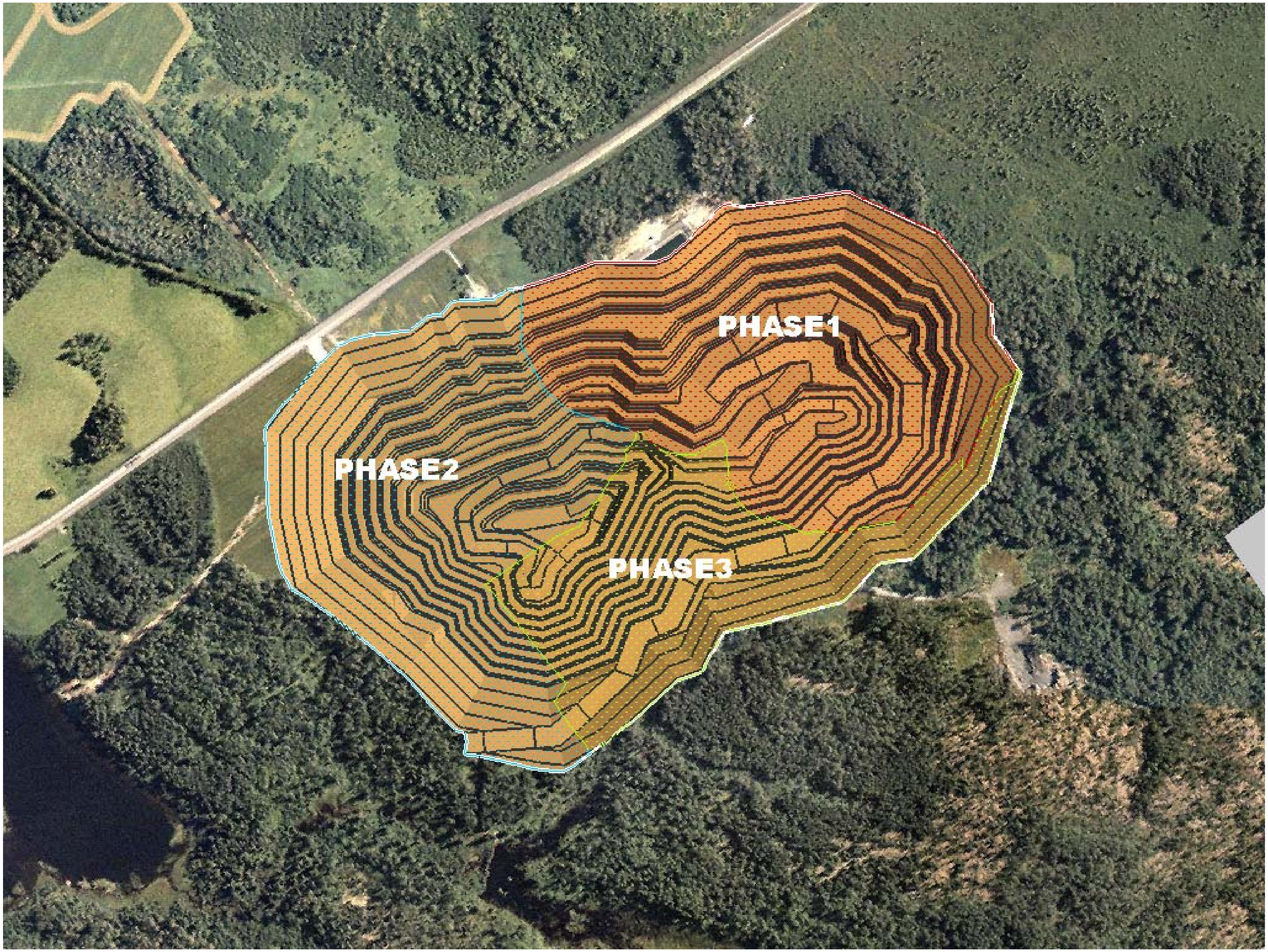

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BLACK FOX

PHASE LAYOUT WITH IMAGE

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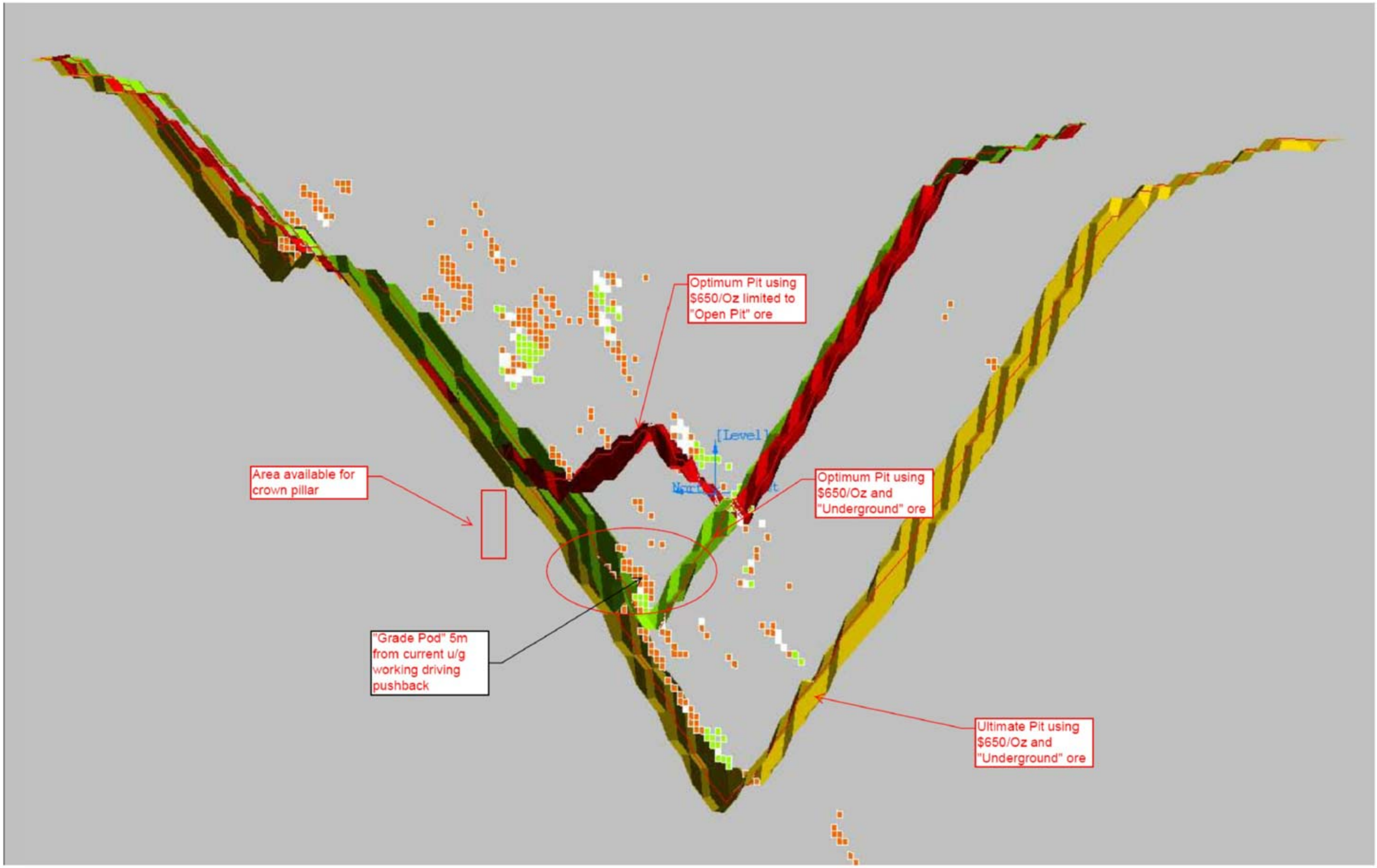

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WHITTLE PIT SHELL RESULTS

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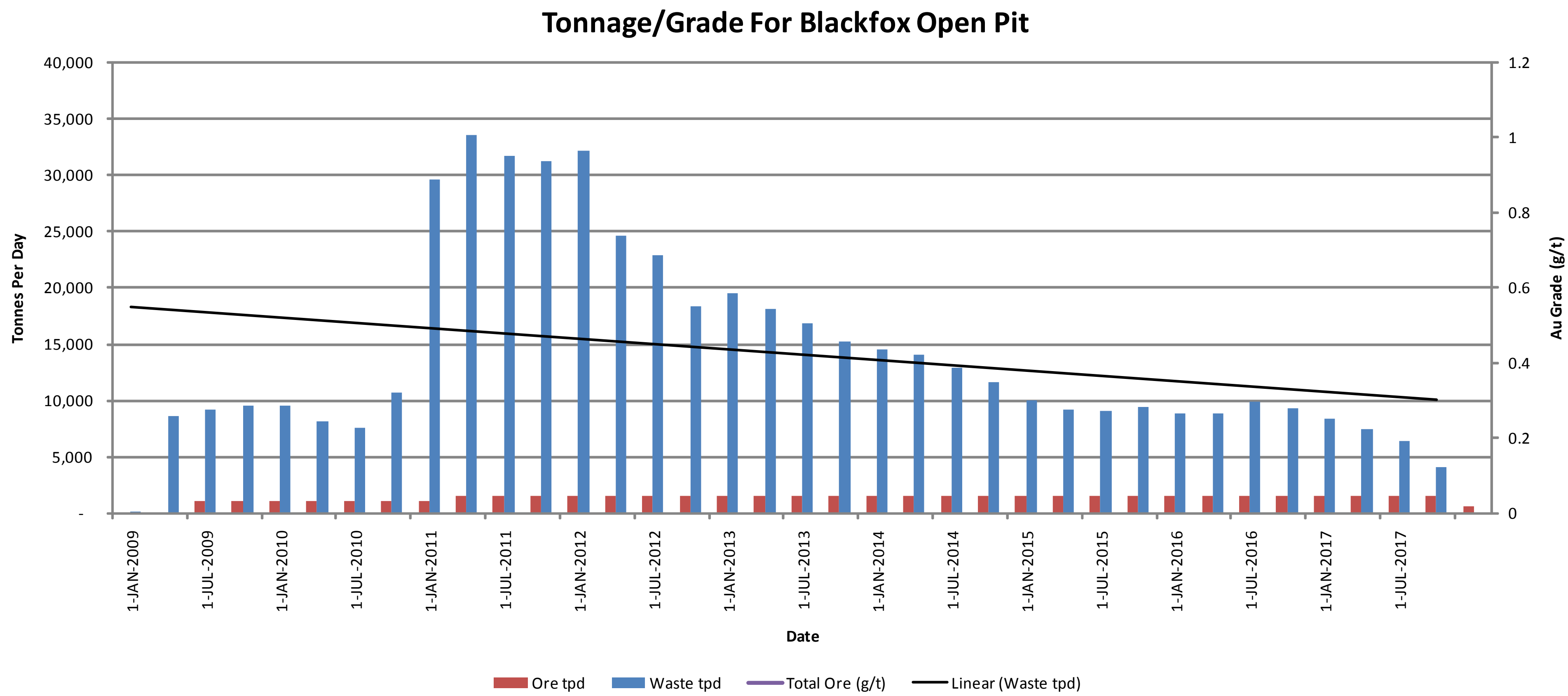

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PRODUCTION SCHEDULE			
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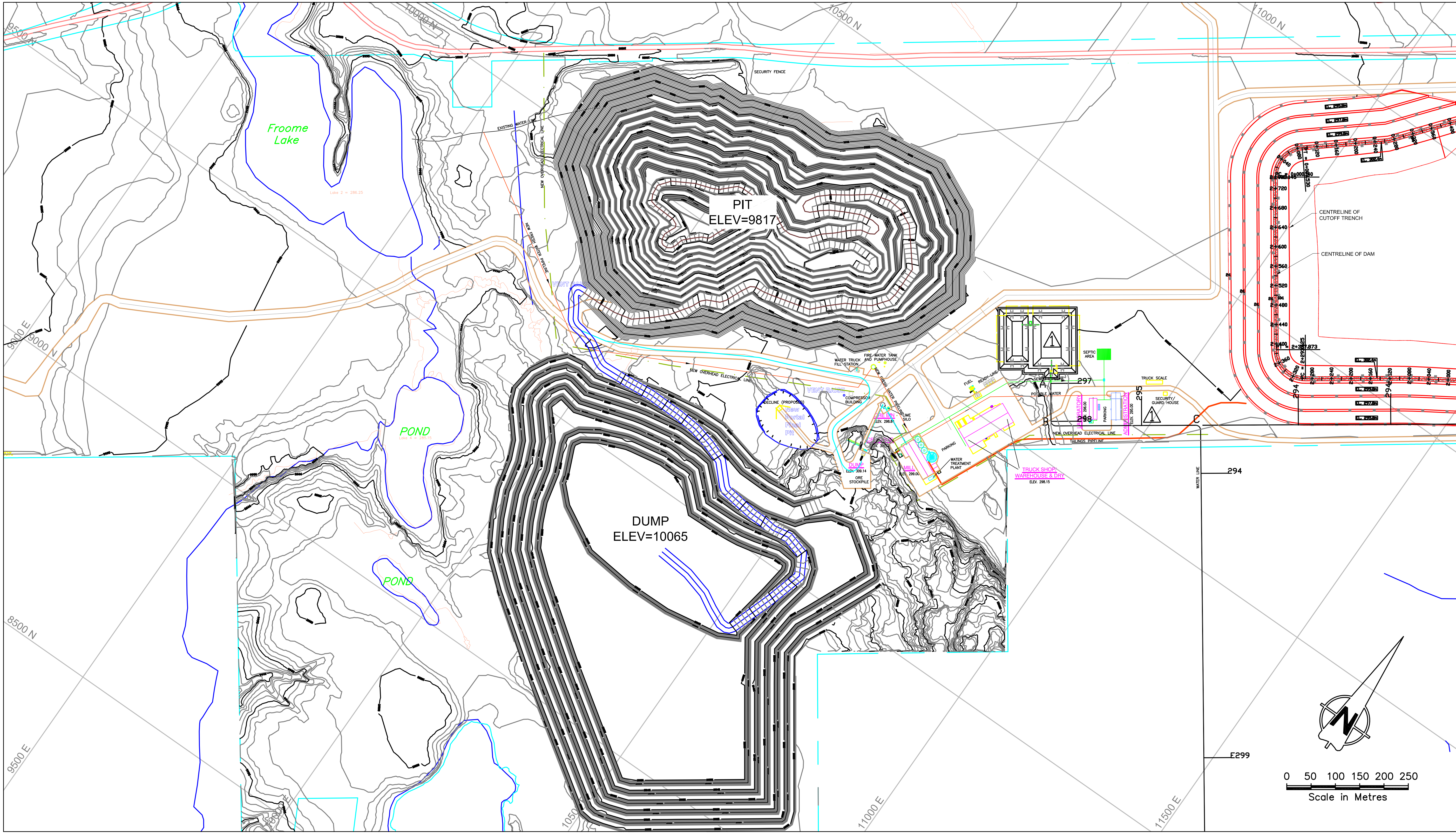

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2017 PHASE DESIGN

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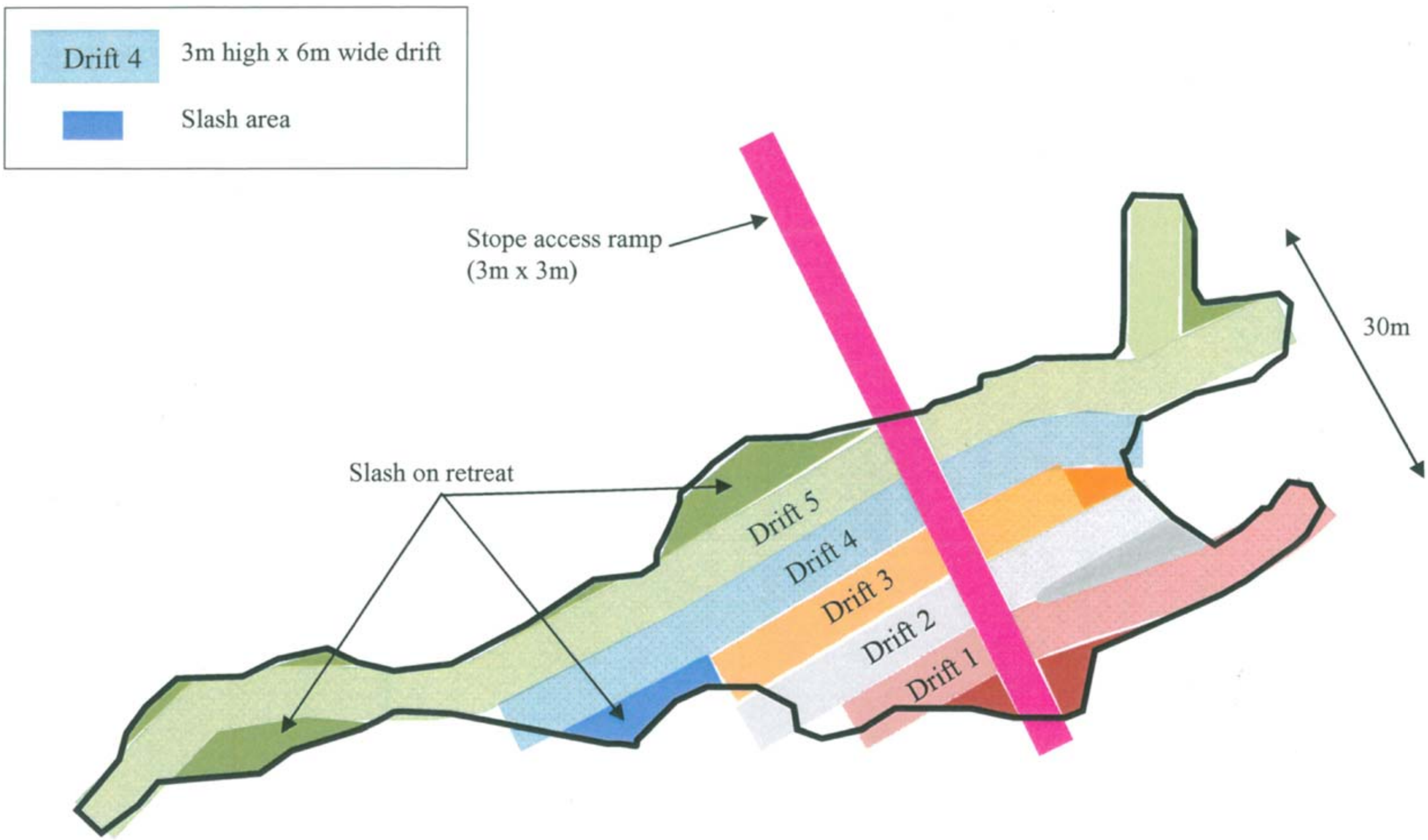

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PLAN VIEW OF TYPICAL CUT AND FILL STOPE LIFT -SHOWING MULTIPLE 6m WIDE DRIFTS

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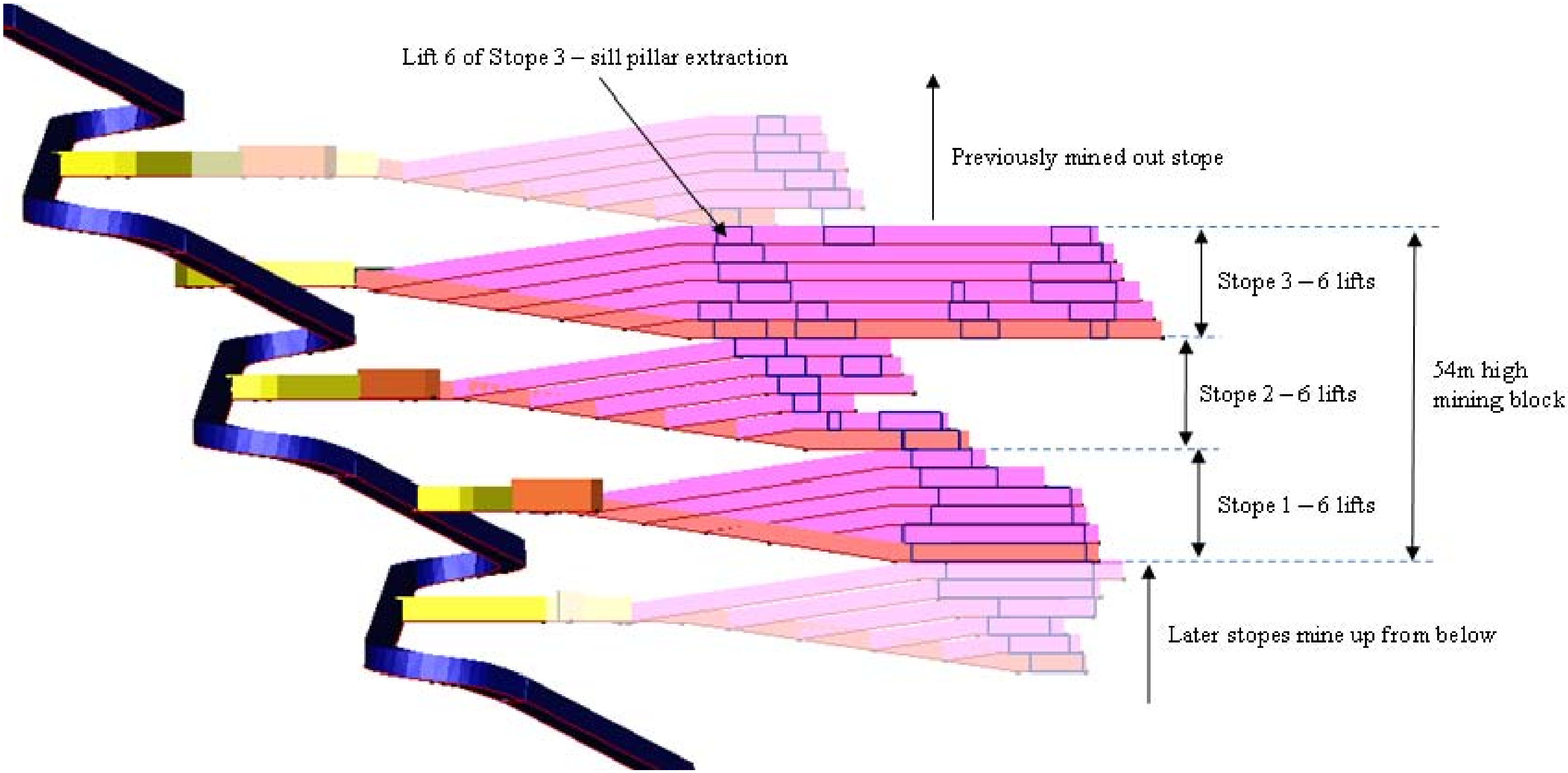

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CUT AND FILL BLOCK, STOPE AND LIFT TYPICAL MINING SEQUENCE

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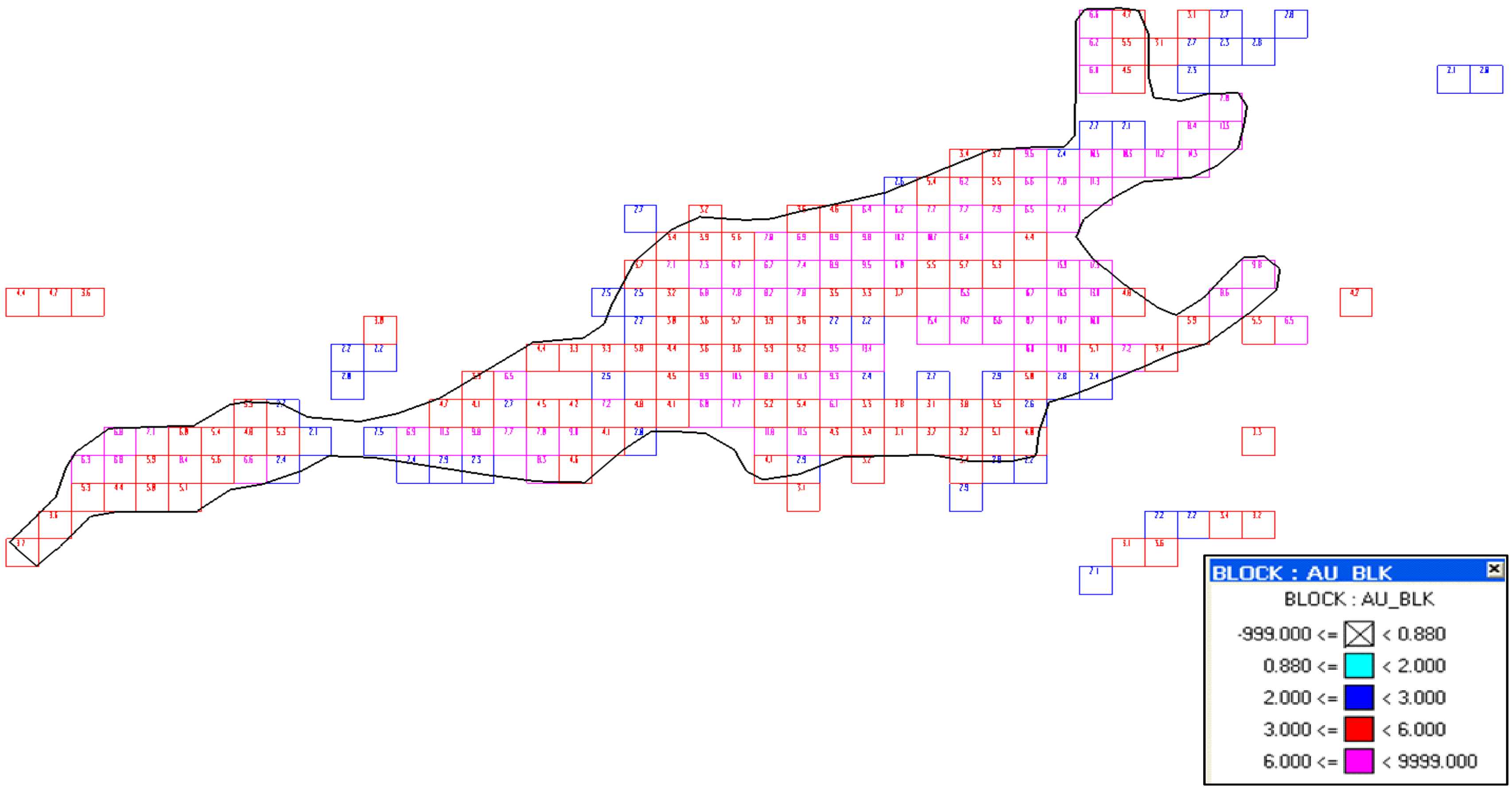

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PLAN VIEW OF TYPICAL CUT AND FILL STOPE LIFT – SHOWING BLOCK MODEL AND STOPE OUTLINE

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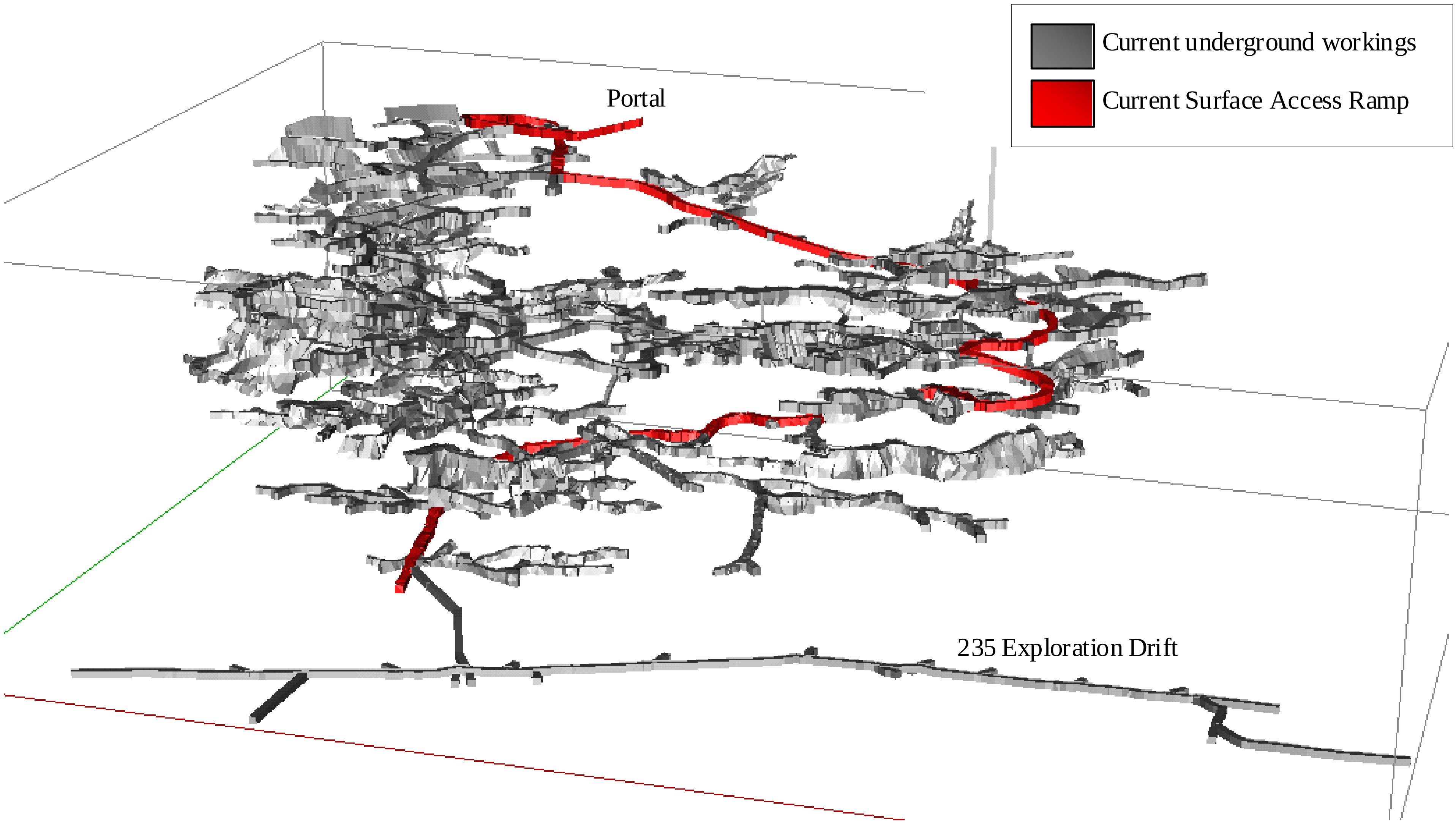

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BLACK FOX	
UNDERGROUND MINE DEVELOPMENT IN 2007 (LOOKING NORTH)	
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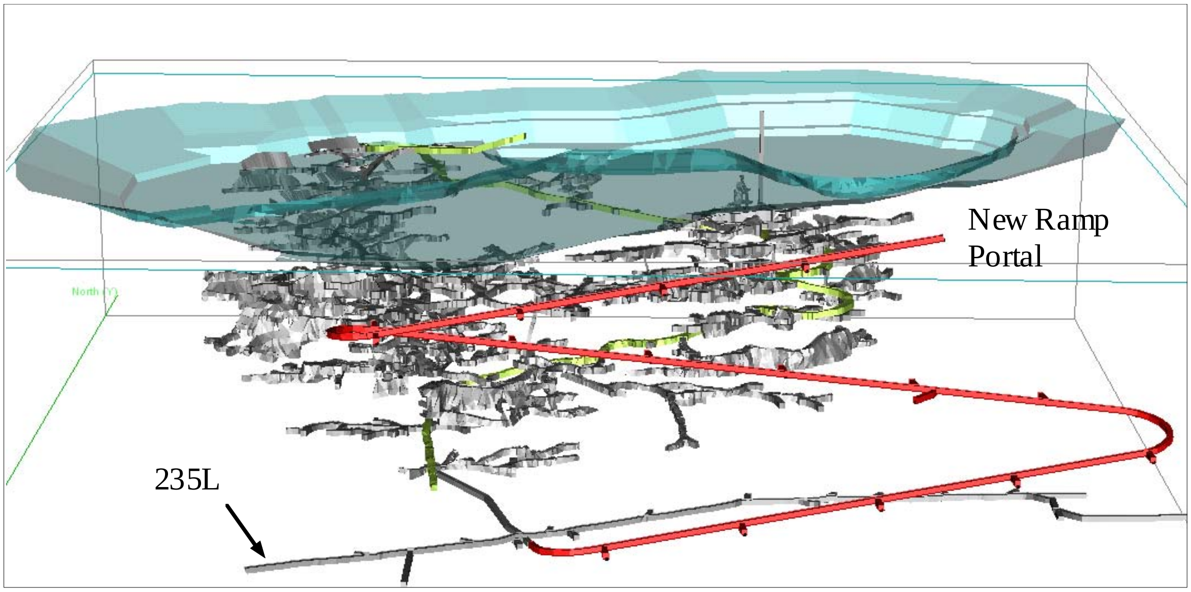
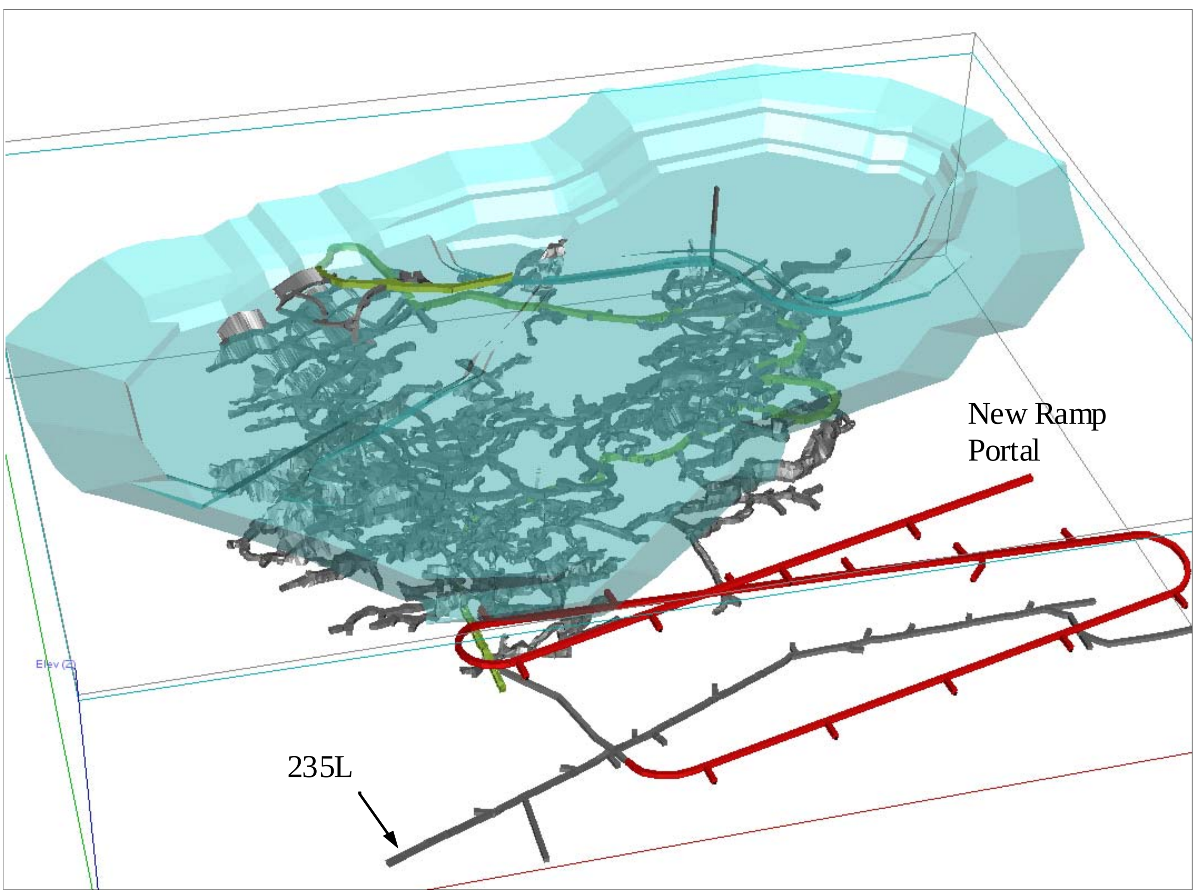


Open Pit Dec 2010
New Surface Access Ramp
Current Surface Access Ramp
Current workings



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NEW SURFACE ACCESS RAMP

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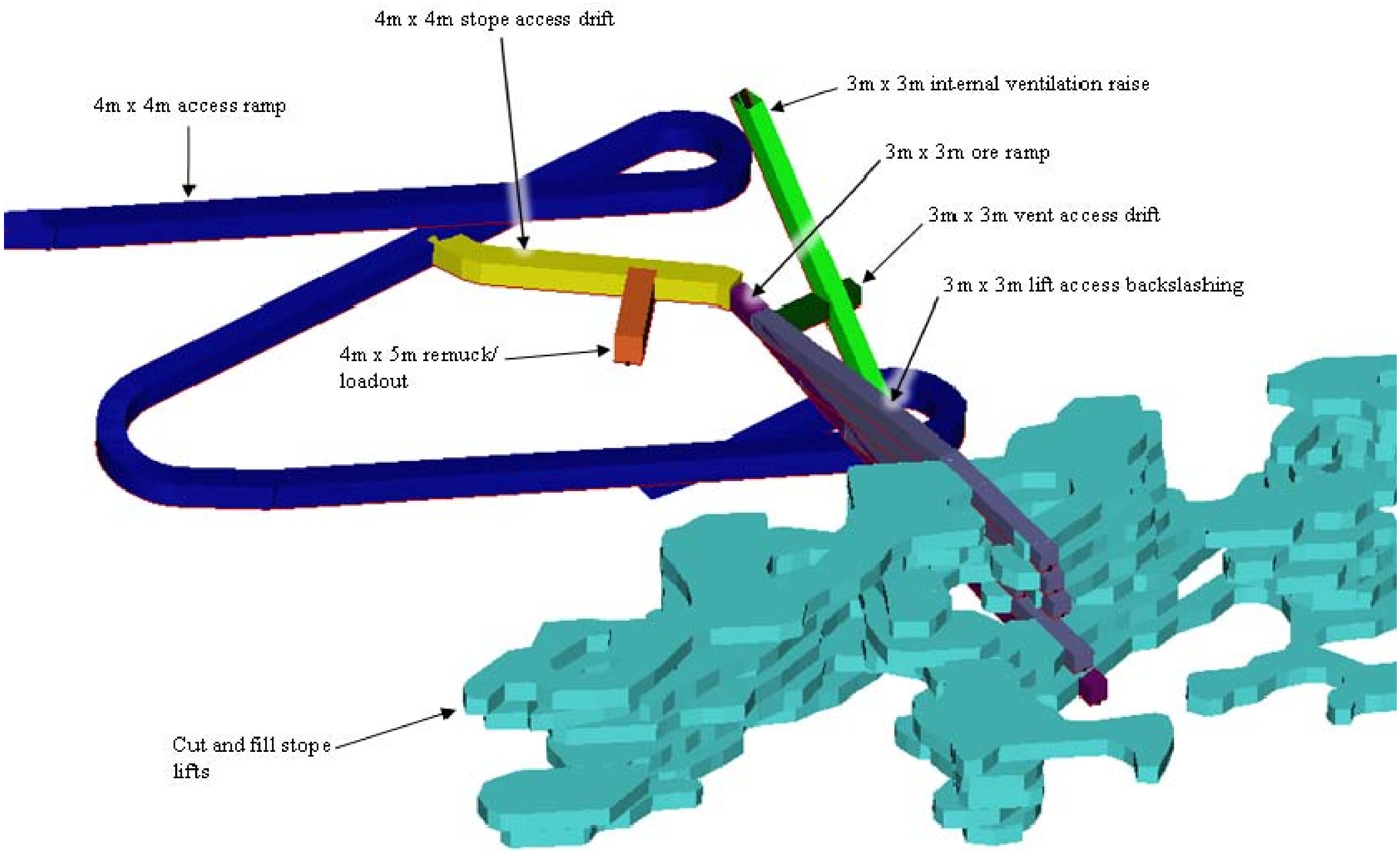

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		TYPICAL CUT AND FILL STOPE AT ACCESS LAYOUT			
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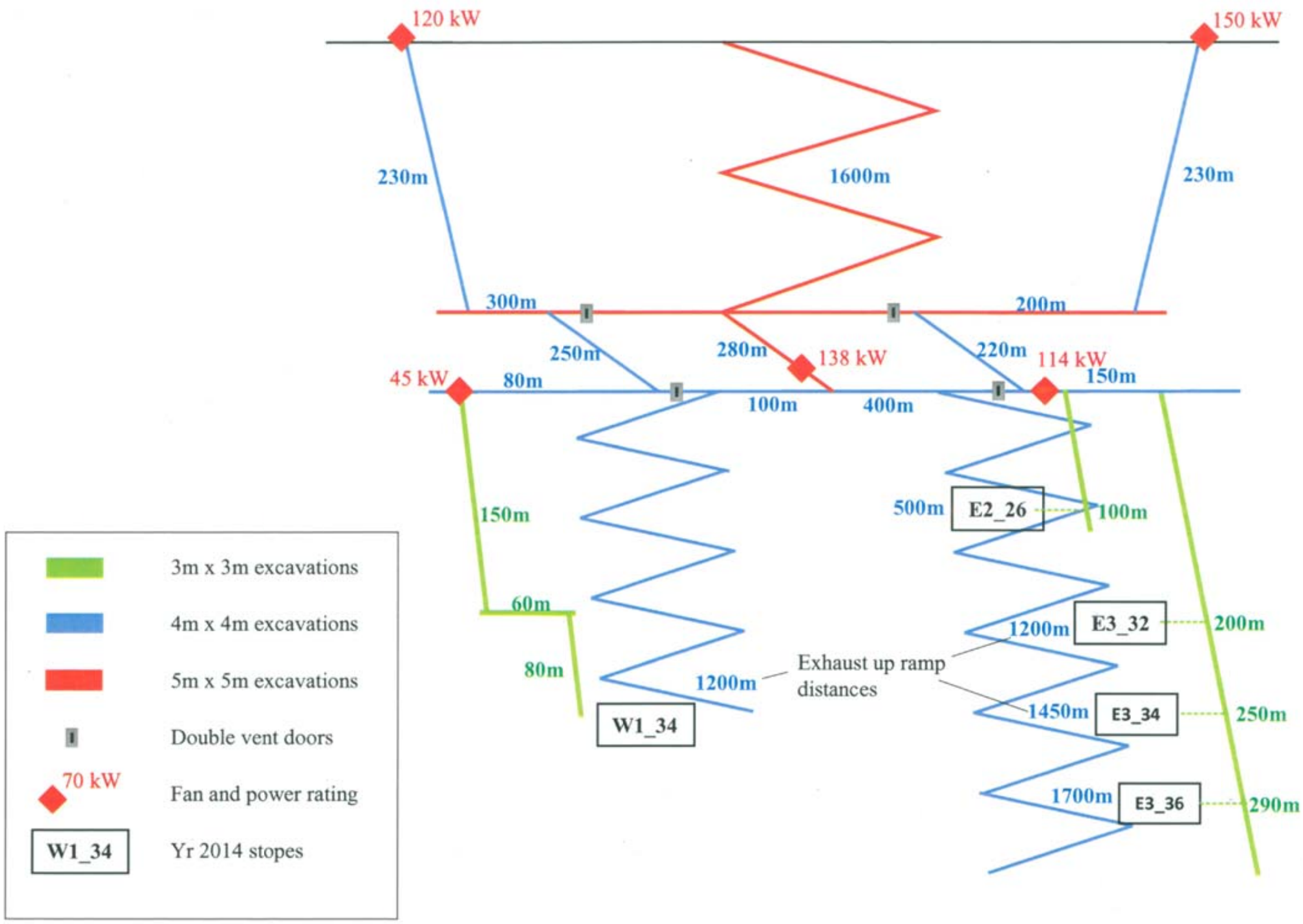

SRK Consulting Engineers and Scientists 7175 West Jefferson Ave. Suite 3000 Denver, Colorado 80235 303-985-1333	ApolloGold	BLACK FOX SCHEMATIC VIEW OF VENTILATION FLOW DIRECTION
SRK JOB NO.: 0144418.00 / TASK 0005 FILE NAME: 0144418.00.Fig.16-11.Schematic.View.Fan.Locations.Drift.Lengths.dwg	TIMMINS, ONTARIO, CANADA	DATE: MAR. 2008 · APPROVED: MR · FIGURE: 16-11 · REVISION NO. A

Exhibit 16.1

Apollo Gold
Black Fox
PRE-TAX CASH FLOW

Till Removal | Stock Mill (Owner) | Holt Mill (Toll) | Till Removal | Black Fox Mill (Owner - Not Used) | END UG | END OP

Description	Units	LoM Total	Q4 -02	2008 Q1 -01	Q2 00	Q3 01	Q4 02	2009 Q1 03	Q2 04	Q3 05	Q4 06	2010 Q1 07	Q2 08	Q3 09	Q4 10
PRODUCTION SUMMARY															
OPEN PIT															
Waste	kt	56,882	0	0	0	900	1,395	2,067	826	862	861	724	2,481	2,795	4,508
Ore	kt	4,350	0	0	0	0.02	0.75	92	90	91	91	89	90	92	137
Total	kt	61,232	0	0	0	900	1,396	2,160	916	953	952	813	2,571	2,887	4,644
s/r	*wst:ore*	*13.1*	*0.0*	*0.0*	*0.0*	*54,991.7*	*1,868.7*	*22.4*	*9.2*	*9.5*	*9.5*	*8.1*	*27.6*	*30.3*	*33.0*
Grade	gpt	5.21	0.00	0.00	0.00	1.56	13.74	13.32	9.76	7.63	5.59	6.19	5.99	5.48	4.77
Contained Gold	koz	729	0	0	0	0	0	40	28	22	16	18	17	16	21
UNDERGROUND															
Total Development	m	35,920	0	0	0	1,109	1,820	1,935	2,400	2,023	2,168	1,520	1,401	1,293	1,650
Ore	kt	2,114	0	0	0	6	12	12	53	75	62	77	82	81	73
Grade	gpt	8.82	0.00	0.00	0.00	15.87	7.51	5.28	12.98	7.60	8.42	9.28	11.99	12.67	11.16
Contained Gold	koz	599	0	0	0	3	3	2	22	18	17	23	32	33	26
MILL															
Ore Treated															
Owner-Stock Mill	kt	3,422	0	0	0	0	0	99	99	99	99	99	99	99	99
Toll- Holt Mill	kt	3,041	0	0	0	0	0	24	44	67	54	67	73	74	111
Owner-Black Fox Mill	kt	0	0	0	0	0	0	0	0	0	0	0	0	0	0
Total	kt	6,463	0	0	0	0	0	123	143	166	153	166	172	173	210
Ore Grade															
Owner-Stock Mill	gpt	6.47	0.00	0.00	0.00	0.00	0.00	12.08	10.96	7.61	6.74	7.62	8.86	8.84	7.00
Toll- Holt Mill	gpt	6.30	0.00	0.00	0.00	0.00	0.00	12.08	10.96	7.61	6.74	7.62	8.86	8.84	7.00
Owner-Black Fox Mill	gpt	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
Total	gpt	6.39	0.00	0.00	0.00	0.00	0.00	12.08	10.96	7.61	6.74	7.62	8.86	8.84	7.00
Contained Gold															
Owner-Stock Mill	koz	712	0	0	0	0	0	38	35	24	21	24	28	28	22
Toll- Holt Mill	koz	616	0	0	0	0	0	10	16	16	12	17	21	21	25
Owner-Black Fox Mill	koz	0	0	0	0	0	0	0	0	0	0	0	0	0	0
Total	koz	1,328	0	0	0	0	0	48	51	41	33	41	49	49	47
Recovered Gold															
Owner-Stock Mill	koz	676	0	0	0	0	0	37	33	23	20	23	27	27	21
Toll- Holt Mill	koz	585	0	0	0	0	0	9	15	16	11	16	20	20	24
Owner-Black Fox Mill	koz	0	0	0	0	0	0	0	0	0	0	0	0	0	0
Total	koz	1,262	0	0	0	0	0	46	48	39	31	39	47	47	45
CASH FLOW															
Gross Revenue															
Payable Gold	99.92%	1,261	0	0	0	0	0	46	48	39	31	39	47	47	45
Gold (Au)	US$/oz	$750	$750	$750	$750	$750	$750	$750	$750	$750	$750	$750	$750	$750	$750
Gross Revenue	US$000	945,455	0	0	0	0	0	34,157	35,992	28,913	23,562	29,045	34,918	35,054	33,553
Refinery															
Refining	$0.55	694	0	0	0	0	0	25	26	21	17	21	26	26	25
Trans., Ins. & Assay	US$000	921	0	0	0	0	0	31	33	28	24	28	32	32	31
Refinery	US$000	1,615	0	0	0	0	0	56	59	49	41	49	57	58	56
NSR	US$000	943,840	0	0	0	0	0	34,100	35,933	28,864	23,520	28,996	34,860	34,997	33,497
Royalty															
Royalty 1	US$000	0	0	0	0	0	0	0	0	0	0	0	0	0	0
Royalty 2	US$000	0	0	0	0	0	0	0	0	0	0	0	0	0	0
Gross Income From Mining	US$000	943,840	0	0	0	0	0	34,100	35,933	28,864	23,520	28,996	34,860	34,997	33,497
Realized Price	US$/oz-Au	$748.72	$0.00	$0.00	$0.00	$0.00	$0.00	$748.76	$748.77	$748.73	$748.69	$748.73	$748.77	$748.77	$748.76
Operating Costs															
Open Pit Mine	US$000	107,348	0	0	0	0	0	1,738	1,796	1,835	1,837	1,804	1,766	2,079	5,002
Underground Mine	US$000	117,083	0	0	0	1,452	1,731	1,876	3,257	3,749	3,556	3,941	4,134	4,340	4,081
Mine G&A	US$000	42,576	0	0	104	559	603	1,065	1,361	1,330	1,269	1,339	1,265	1,266	1,272
Holt Mill (Toll)	US$000	120,256	0	0	0	0	0	1,006	1,782	2,655	2,145	2,680	2,907	2,944	4,353
Stock Mill	US$000	84,265	0	0	0	0	0	2,421	2,421	2,422	2,422	2,428	2,428	2,428	2,428
Black Fox Mill	US$000	0	0	0	0	0	0	0	0	0	0	0	0	0	0
G&A	US$000	19,907	0	0	0	779	504	614	614	544	544	544	544	544	544
Operating Costs	US$000	485,704						8,719	11,231	12,535	11,774	12,736	13,044	13,601	17,679
per oz-Au	(US$/oz)	385.29	0.00	0.00	0.00	0.00	0.00	191.45	234.03	325.16	374.77	328.86	280.17	291.01	395.16
per milled ton	(US$/t)	75.15	0.00	0.00	0.00	0.00	0.00	70.61	78.28	75.54	77.05	76.52	75.72	78.52	84.38
Total Cash Costs															
Refining	US$000	1,615	0	0	0	0	0	56	59	49	41	49	57	58	56
Royalty	US$000	0	0	0	0	0	0	0	0	0	0	0	0	0	0
Operating	US$000	485,704	0	0	0	0	0	8,719	11,231	12,535	11,774	12,736	13,044	13,601	17,679
Total Cash Cost	US$000	487,319	0	0	0	0	0	8,775	11,290	12,584	11,815	12,785	13,101	13,659	17,734
per oz-Au	(US$/oz)	386.57	0.00	0.00	0.00	0.00	0.00	192.69	235.26	326.42	376.08	330.12	281.41	292.24	396.40
per milled ton	(US$/t)	75.40	0.00	0.00	0.00	0.00	0.00	71.06	78.69	75.84	77.32	76.82	76.05	78.85	84.65
Cash Operating Margin	US$000	458,136	0	0	0	0	0	25,381	24,702	16,329	11,747	16,260	21,816	21,395	15,819
per oz-Au	(US$/oz)	363.43	0.00	0.00	0.00	0.00	0.00	557.31	514.74	423.58	373.92	419.88	468.59	457.76	353.60
per milled ton	(US$/t)	70.88	0.00	0.00	0.00	0.00	0.00	205.54	172.18	98.41	76.88	97.70	126.64	123.51	75.50

Description	Units	2011 Q1 11	Q2 12	Q3 13	Q4 14	2012 Q1 15	Q2 16	Q3 17	Q4 18	2013 Q1 19	Q2 20	Q3 21	Q4 22	2014 Q1 23	Q2 24	Q3 25	Q4 26
PRODUCTION SUMMARY																	
OPEN PIT																	
Waste	kt	4,766	2,871	2,834	2,923	2,210	2,053	1,663	1,768	1,612	1,515	1,379	1,322	1,251	1,163	1,055	903
Ore	kt	134	135	137	137	135	135	137	136	133	135	137	136	133	135	136	136
Total	kt	4,899	3,006	2,970	3,060	2,345	2,188	1,799	1,904	1,746	1,650	1,515	1,459	1,385	1,298	1,191	1,039
s/r	*wst:ore*	*35.7*	*21.3*	*20.8*	*21.4*	*16.4*	*15.2*	*12.2*	*12.9*	*12.1*	*11.2*	*10.1*	*9.7*	*9.4*	*8.6*	*7.7*	*6.6*
Grade	gpt	4.79	4.29	5.39	5.13	4.03	3.72	3.49	3.36	3.50	3.75	4.82	5.45	6.02	4.29	4.32	4.36
Contained Gold	koz	21	19	24	22	17	16	15	15	15	16	21	24	26	19	19	19
UNDERGROUND																	
Total Development	m	1,475	1,381	1,223	1,202	849	835	1,007	845	1,139	854	725	933	1,546	1,011	1,134	648
Ore	kt	90	106	87	73	88	88	91	67	87	86	96	89	98	101	79	80
Grade	gpt	10.73	11.58	12.99	10.58	7.50	7.79	8.80	11.42	6.88	6.63	4.84	6.35	5.65	7.38	7.01	7.18
Contained Gold	koz	31	39	36	25	21	22	26	25	19	18	15	18	18	24	18	18
MILL																	
Ore Treated																	
Owner-Stock Mill	kt	99	99	99	99	99	99	99	99	99	99	99	99	99	99	99	99
Toll- Holt Mill	kt	124	142	124	111	124	124	129	105	121	122	134	126	133	137	117	117
Owner-Black Fox Mill	kt	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
Total	kt	223	241	223	210	223	223	228	204	220	221	233	225	232	236	216	216
Ore Grade																	
Owner-Stock Mill	gpt	7.18	7.50	8.34	7.03	5.40	5.33	5.62	6.02	4.83	4.87	4.83	5.80	5.86	5.61	5.31	5.40
Toll- Holt Mill	gpt	7.18	7.50	8.34	7.03	5.40	5.33	5.62	6.02	4.83	4.87	4.83	5.80	5.86	5.61	5.31	5.40
Owner-Black Fox Mill	gpt	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
Total	gpt	7.18	7.50	8.34	7.03	5.40	5.33	5.62	6.02	4.83	4.87	4.83	5.80	5.86	5.61	5.31	5.40
Contained Gold																	
Owner-Stock Mill	koz	23	24	27	22	17	17	18	19	15	16	15	18	19	18	17	17
Toll- Holt Mill	koz	29	34	33	25	22	21	23	20	19	19	21	24	25	25	20	20
Owner-Black Fox Mill	koz	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
Total	koz	51	58	60	47	39	38	41	39	34	35	36	42	44	43	37	38
Recovered Gold																	
Owner-Stock Mill	koz	22	23	25	21	16	16	17	18	15	15	15	18	18	17	16	16
Toll- Holt Mill	koz	27	33	32	24	20	20	22	19	18	18	20	22	24	23	19	19
Owner-Black Fox Mill	koz	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
Total	koz	49	55	57	45	37	36	39	37	32	33	34	40	42	40	35	36
CASH FLOW																	
Gross Revenue																	
Payable Gold	99.92%	49	55	57	45	37	36	39	37	32	33	34	40	41	40	35	36
Gold (Au)	US$/oz	$750	$750	$750	$750	$750	$750	$750	$750	$750	$750	$750	$750	$750	$750	$750	$750
Gross Revenue	US$000	36,661	41,367	42,604	33,742	27,567	27,202	29,288	28,060	24,325	24,626	25,721	29,950	31,124	30,285	26,231	26,722
Refinery																	
Refining	$0.55	27	30	31	25	20	20	21	21	18	18	19	22	23	22	19	20
Trans., Ins. & Assay	US$000	33	36	37	31	27	26	28	27	24	25	25	28	29	29	26	26
Refinery	US$000	60	67	69	56	47	46	49	48	42	43	44	50	52	51	45	46
NSR	US$000	36,601	41,300	42,535	33,686	27,520	27,155	29,238	28,012	24,283	24,584	25,677	29,900	31,072	30,234	26,186	26,676
Royalty																	
Royalty 1	US$000	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
Royalty 2	US$000	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
Gross Income From Mining	US$000	36,601	41,300	42,535	33,686	27,520	27,155	29,238	28,012	24,283	24,584	25,677	29,900	31,072	30,234	26,186	26,676
Realized Price	US$/oz-Au	$748.77	$748.79	$748.79	$748.76	$748.72	$748.72	$748.73	$748.73	$748.70	$748.70	$748.71	$748.74	$748.75	$748.74	$748.71	$748.72
Operating Costs																	
Open Pit Mine	US$000	5,480	5,372	5,359	5,454	4,466	4,048	3,545	3,797	3,651	3,550	3,312	3,259	3,184	3,086	2,955	2,610
Underground Mine	US$000	4,585	5,037	4,526	4,004	4,596	4,626	4,640	4,074	4,465	4,463	4,785	4,666	4,807	4,854	4,391	4,194
Mine G&A	US$000	1,336	1,390	1,322	1,277	1,315	1,313	1,316	1,234	1,299	1,295	1,328	1,302	1,333	1,340	1,263	1,261
Holt Mill (Toll)	US$000	4,883	5,578	4,882	4,361	4,878	4,884	5,064	4,126	4,763	4,795	5,252	4,974	5,225	5,371	4,604	4,613
Stock Mill	US$000	2,427	2,428	2,429	2,428	2,428	2,428	2,428	2,428	2,428	2,428	2,428	2,429	2,428	2,428	2,430	2,429
Black Fox Mill	US$000	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
G&A	US$000	524	524	524	524	524	524	524	524	524	524	524	524	524	524	524	524
Operating Costs	US$000	19,234	20,329	19,041	18,047	18,207	17,822	17,517	16,183	17,129	17,054	17,629	17,153	17,502	17,602	16,167	15,631
per oz-Au	(US$/oz)	393.49	368.57	335.21	401.15	495.34	491.39	448.58	432.54	528.13	519.40	514.05	429.54	421.74	435.91	462.26	438.70
per milled ton	(US$/t)	86.19	84.34	85.35	86.07	81.63	79.86	76.90	79.46	77.84	77.21	75.77	76.07	75.45	74.67	74.87	72.31
Total Cash Costs																	
Refining	US$000	60	67	69	56	47	46	49	48	42	43	44	50	52	51	45	46
Royalty	US$000	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
Operating	US$000	19,234	20,329	19,041	18,047	18,207	17,822	17,517	16,183	17,129	17,054	17,629	17,153	17,502	17,602	16,167	15,631
Total Cash Cost	US$000	19,294	20,396	19,110	18,103	18,254	17,869	17,567	16,230	17,172	17,097	17,673	17,203	17,554	17,652	16,212	15,677
per oz-Au	(US$/oz)	394.71	369.78	336.41	402.39	496.62	492.67	449.85	433.81	529.44	520.70	515.34	430.80	422.99	437.16	463.54	439.99
per milled ton	(US$/t)	86.45	84.61	85.66	86.33	81.84	80.07	77.11	79.69	78.03	77.41	75.96	76.30	75.67	74.89	75.08	72.52
Cash Operating Margin	US$000	17,367	20,971	23,494	15,639	9,313	9,333	11,721	11,830	7,154	7,529	8,047	12,747	13,570	12,632	10,019	11,046
per oz-Au	(US$/oz)	355.29	380.22	413.59	347.61	253.38	257.33	300.15	316.19	220.56	229.30	234.66	319.20	327.01	312.84	286.46	310.01
per milled ton	(US$/t)	77.82	87.00	105.31	74.58	41.76	41.82	51.45	58.08	32.51	34.09	34.59	56.53	58.50	53.59	46.40	51.10

Description	Units	2015 Q1 27	Q2 28	Q3 29	Q4 30	2016 Q1 31	Q2 32	Q3 33	Q4 34	2017 Q1 35	Q2 36	Q3 37	Q4 38	2018 Q1 39	Q2 40	Q3 41	Q4 42
PRODUCTION SUMMARY																	
OPEN PIT																	
Waste	kt	819	811	855	800	793	886	844	759	659	576	373	0	0	0	0	0
Ore	kt	134	135	136	137	135	135	136	136	133	135	56	0	0	0	0	0
Total	kt	953	946	992	936	928	1,021	980	896	792	711	429	0	0	0	0	0
s/r	*wst:ore*	*6.1*	*6.0*	*6.3*	*5.9*	*5.9*	*6.6*	*6.2*	*5.6*	*4.9*	*4.3*	*6.6*	*0.0*	*0.0*	*0.0*	*0.0*	*0.0*
Grade	gpt	5.06	5.29	5.58	5.74	5.57	4.99	5.43	6.06	4.97	4.81	4.86	0.00	0.00	0.00	0.00	0.00
Contained Gold	koz	22	23	24	25	24	22	24	27	21	21	9	0	0	0	0	0
UNDERGROUND																	
Total Development	m	594	663	266	271	0	0	0	0	0	0	0	0	0	0	0	0
Ore	kt	63	39	37	26	9	0	0	0	0	0	0	0	0	0	0	0
Grade	gpt	9.12	8.63	8.12	6.31	6.78	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
Contained Gold	koz	18	11	10	5	2	0	0	0	0	0	0	0	0	0	0	0
MILL																	
Ore Treated																	
Owner-Stock Mill	kt	99	99	99	99	99	99	99	99	99	99	56	0	0	0	0	0
Toll- Holt Mill	kt	97	75	74	64	45	36	37	37	34	36	0	0	0	0	0	0
Owner-Black Fox Mill	kt	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
Total	kt	196	174	173	163	144	135	136	136	133	135	56	0	0	0	0	0
Ore Grade																	
Owner-Stock Mill	gpt	6.36	6.04	6.12	5.83	5.65	4.99	5.43	6.06	4.97	4.81	4.86	0.00	0.00	0.00	0.00	0.00
Toll- Holt Mill	gpt	6.36	6.04	6.12	5.83	5.65	4.99	5.43	6.06	4.97	4.81	4.86	0.00	0.00	0.00	0.00	0.00
Owner-Black Fox Mill	gpt	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
Total	gpt	6.36	6.04	6.12	5.83	5.65	4.99	5.43	6.06	4.97	4.81	4.86	0.00	0.00	0.00	0.00	0.00
Contained Gold																	
Owner-Stock Mill	koz	20	19	19	19	18	16	17	19	16	15	9	0	0	0	0	0
Toll- Holt Mill	koz	20	15	15	12	8	6	7	7	6	6	0	0	0	0	0	0
Owner-Black Fox Mill	koz	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
Total	koz	40	34	34	31	26	22	24	27	21	21	9	0	0	0	0	0
Recovered Gold																	
Owner-Stock Mill	koz	19	18	18	18	17	15	16	18	15	15	8	0	0	0	0	0
Toll- Holt Mill	koz	19	14	14	11	8	5	6	7	5	5	0	0	0	0	0	0
Owner-Black Fox Mill	koz	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
Total	koz	38	32	32	29	25	21	23	25	20	20	8	0	0	0	0	0
CASH FLOW																	
Gross Revenue																	
Payable Gold	99.92%	38	32	32	29	25	21	23	25	20	20	8	0	0	0	0	0
Gold (Au)	US$/oz	$750	$750	$750	$750	$750	$750	$750	$750	$750	$750	$750	$750	$750	$750	$750	$750
Gross Revenue	US$000	28,541	24,021	24,253	21,742	18,590	15,421	16,957	18,926	15,199	14,875	6,262	0	0	0	0	0
Refinery																	
Refining	$0.55	21	18	18	16	14	11	12	14	11	11	5	0	0	0	0	0
Trans., Ins. & Assay	US$000	27	24	24	23	20	18	19	21	18	18	12	0	0	0	0	0
Refinery	US$000	48	42	42	39	34	30	32	35	29	29	16	0	0	0	0	0
NSR	US$000	28,493	23,979	24,211	21,703	18,556	15,391	16,926	18,891	15,170	14,846	6,246	0	0	0	0	0
Royalty																	
Royalty 1	US$000	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
Royalty 2	US$000	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
Gross Income From Mining	US$000	28,493	23,979	24,211	21,703	18,556	15,391	16,926	18,891	15,170	14,846	6,246	0	0	0	0	0
Realized Price	US$/oz-Au	$748.73	$748.69	$748.70	$748.67	$748.63	$748.56	$748.60	$748.63	$748.56	$748.55	$748.04	$0.00	$0.00	$0.00	$0.00	$0.00
Operating Costs																	
Open Pit Mine	US$000	2,517	2,515	2,613	2,566	2,570	2,741	2,675	2,501	2,126	2,019	1,522	0	0	0	0	0
Underground Mine	US$000	3,749	3,245	2,208	1,797	1,253	0	0	0	0	0	0	0	0	0	0	0
Mine G&A	US$000	1,273	1,258	1,243	1,169	1,045	709	708	707	705	703	698	0	0	0	0	0
Holt Mill (Toll)	US$000	3,834	2,970	2,944	2,545	1,804	1,460	1,519	1,519	1,402	1,460	74	0	0	0	0	0
Stock Mill	US$000	2,428	2,428	2,430	2,429	2,428	2,428	2,429	2,429	2,428	2,428	1,728	0	0	0	0	0
Black Fox Mill	US$000	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
G&A	US$000	524	524	524	524	524	524	524	524	524	524	524	0	0	0	0	0
Operating Costs	US$000	14,325	12,940	11,961	11,031	9,624	7,862	7,854	7,679	7,185	7,134	4,545	0	0	0	0	0
per oz-Au	(US$/oz)	376.42	404.02	369.87	380.52	388.24	382.36	347.36	304.32	354.53	359.69	544.33	0.00	0.00	0.00	0.00	0.00
per milled ton	(US$/t)	73.03	74.42	69.06	67.70	66.90	58.23	57.54	56.26	53.82	52.84	80.72	0.00	0.00	0.00	0.00	0.00
Total Cash Costs																	
Refining	US$000	48	42	42	39	34	30	32	35	29	29	16	0	0	0	0	0
Royalty	US$000	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
Operating	US$000	14,325	12,940	11,961	11,031	9,624	7,862	7,854	7,679	7,185	7,134	4,545	0	0	0	0	0
Total Cash Cost	US$000	14,373	12,982	12,003	11,069	9,658	7,891	7,886	7,714	7,214	7,163	4,561	0	0	0	0	0
per oz-Au	(US$/oz)	377.69	405.33	371.18	381.85	389.62	383.80	348.77	305.69	355.97	361.14	546.29	0.00	0.00	0.00	0.00	0.00
per milled ton	(US$/t)	73.28	74.66	69.30	67.94	67.13	58.45	57.77	56.51	54.04	53.06	81.01	0.00	0.00	0.00	0.00	0.00
Cash Operating Margin	US$000	14,168	11,039	12,250	10,672	8,933	7,529	9,072	11,212	7,985	7,712	1,701	0	0	0	0	0
per oz-Au	(US$/oz)	372.31	344.67	378.82	368.15	360.38	366.20	401.23	444.31	394.03	388.86	203.71	0.00	0.00	0.00	0.00	0.00
per milled ton	(US$/t)	72.23	63.49	70.73	65.50	62.10	55.77	66.46	82.14	59.81	57.13	30.21	0.00	0.00	0.00	0.00	0.00

Exhibit 16.1

Apollo Gold
Black Fox
PRE-TAX CASH FLOW

Till Removal | Till Removal | Stock Mill (Owner) | Holt Mill (Toll) | Black Fox Mill (Owner - Not Used) | END UG | END OP

Description	Units	LoM Total	Q4 -02	2008 Q1 -01	Q2 00	Q3 01	Q4 02	2009 Q1 03	Q2 04	Q3 05	Q4 06	2010 Q1 07	Q2 08	Q3 09	Q4 10	2011 Q1 11	Q2 12	Q3 13	Q4 14
CASH FLOW (Continued)																			
Capital Costs																			
Open Pit Mine	US$000	13,462	0	0	0	0	13,462	0	0	0	0	0	0	0	0	0	0	0	0
Underground Mine	US$000	12,555	0	0	0	5,992	1,738	1,211	3,443	173	0	0	0	0	0	0	0	0	0
Stock Mill & Tailings	US$000	4,251	0	0	0	1,409	1,409	0	0	0	0	0	0	0	0	0	0	0	0
Black Fox Mill	US$000	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
Infrastructure	US$000	24,199	0	0	0	12,099	10,889	1,210	0	0	0	0	0	0	0	0	0	0	0
Bridge to Waste Dump	US$000	446	0	0	0	0	0	0	0	0	0	446	0	0	0	0	0	0	0
Black FoxTailings Dam	US$000	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
Closure Costs	US$000	16,091	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
Owner Costs	US$000	27,116	0	0	345	20,714	4,721	333	333	333	333	0	0	0	0	0	0	0	0
Capital Costs	**US$000**	**98,118**	**0**	**0**	**345**	**40,214**	**32,218**	**2,754**	**3,776**	**506**	**333**	**446**	**0**	**0**	**0**	**0**	**0**	**0**	**0**
							72,777												
Capitalized																			
Capitalized Operating	US$000	5,732	0	0	104	2,790	2,838												
Till Removal	US$000	20,061	0	0	0	1,584	2,429	2,296	0	0	0	0	3,420	3,458	3,458	3,382	34	0	0
Mine Development	US$000	32,171	0	0	0	1,521	2,876	2,330	2,047	1,680	1,951	1,189	1,089	1,103	1,299	1,004	924	1,087	879
Capitalized	**US$000**	**57,963**	**0**	**0**	**104**	**5,895**	**8,143**	**4,625**	**2,047**	**1,680**	**1,951**	**1,189**	**4,509**	**4,561**	**4,757**	**4,386**	**958**	**1,087**	**879**
							14,142												
Working Capital																			
Receivables	30		0	0	0	0	0	2,807	2,958	2,376	1,937	2,387	2,870	2,881	2,758	3,013	3,400	3,502	2,773
Payables	90		0	0	0	0	0	(2,164)	(2,784)	(3,103)	(2,913)	(3,152)	(3,230)	(3,368)	(4,373)	(4,757)	(5,029)	(4,712)	(4,464)
Working Capital	**US$000**		**0**	**0**	**0**	**0**	**0**	**(644)**	**(174)**	**726**	**977**	**765**	**361**	**487**	**1,615**	**1,744**	**1,629**	**1,210**	**1,690**
Change in Working Capital	US$000	0	0	0	0	0	0	644	(469)	(901)	(250)	212	405	(126)	(1,128)	(129)	115	419	(480)
TOTAL CAPITAL	**US$000**	**156,081**	**0**	**0**	**449**	**46,109**	**40,361**	**8,023**	**5,354**	**1,285**	**2,034**	**1,847**	**4,914**	**4,435**	**3,629**	**4,257**	**1,073**	**1,505**	**399**
							86,919												
Gross Income	**US$000**	**943,840**	**0**	**0**	**0**	**0**	**0**	**34,100**	**35,933**	**28,864**	**23,520**	**28,996**	**34,860**	**34,997**	**33,497**	**36,601**	**41,300**	**42,535**	**33,686**
Costs																			
Operating	US$000	485,704	0	0	0	0	0	8,719	11,231	12,535	11,774	12,736	13,044	13,601	17,679	19,234	20,329	19,041	18,047
Capital	US$000	156,081	0	0	449	46,109	40,361	7,379	5,823	2,185	2,285	1,635	4,509	4,561	4,757	4,386	958	1,087	879
Working Capital	US$000	0	0	0	0	0	0	644	(469)	(901)	(250)	212	405	(126)	(1,128)	(129)	115	419	(480)
Total Costs	**US$000**	**641,785**	**0**	**0**	**449**	**46,109**	**40,361**	**16,742**	**16,585**	**13,819**	**13,808**	**14,582**	**17,958**	**18,036**	**21,307**	**23,491**	**21,402**	**20,547**	**18,446**
Cumulative	US$000		0	0	449	46,558	86,919	103,661	120,246	134,065	147,873	162,455	180,413	198,450	219,757	243,248	264,650	285,196	303,643
CASH FLOW	**US$000**	**302,055**	**0**	**0**	**(449)**	**(46,109)**	**(40,361)**	**17,358**	**19,348**	**15,044**	**9,712**	**14,414**	**16,903**	**16,960**	**12,190**	**13,110**	**19,898**	**21,989**	**15,240**
Cumulative	US$000		0	0	(449)	(46,558)	(86,919)	(69,561)	(50,213)	(35,168)	(25,456)	(11,042)	5,860	22,821	35,011	48,121	68,019	90,008	105,248
Present Value	5.0%	227,081	0	0	(438)	(44,423)	(38,404)	16,313	17,959	13,791	8,794	12,889	14,928	14,794	10,502	11,155	16,722	18,251	12,493
NPV			0	0	(438)	(44,861)	(83,265)	(66,952)	(48,994)	(35,202)	(26,408)	(13,519)	1,409	16,203	26,705	37,860	54,581	72,832	85,325
IRR	**%**	**62%**																	

Description	2012 Q1 15	Q2 16	Q3 17	Q4 18	2013 Q1 19	Q2 20	Q3 21	Q4 22	2014 Q1 23	Q2 24	Q3 25	Q4 26
CASH FLOW (Continued)												
Capital Costs												
Open Pit Mine	0	0	0	0	0	0	0	0	0	0	0	0
Underground Mine	0	0	0	0	0	0	0	0	0	0	0	0
Stock Mill & Tailings	0	0	0	0	0	0	0	0	0	1,432	0	0
Black Fox Mill	0	0	0	0	0	0	0	0	0	0	0	0
Infrastructure	0	0	0	0	0	0	0	0	0	0	0	0
Bridge to Waste Dump	0	0	0	0	0	0	0	0	0	0	0	0
Black FoxTailings Dam	0	0	0	0	0	0	0	0	0	0	0	0
Closure Costs	0	0	0	0	0	0	0	0	0	0	0	0
Owner Costs	0	0	0	0	0	0	0	0	0	0	0	0
Capital Costs	**0**	**0**	**0**	**0**	**0**	**0**	**0**	**0**	**0**	**1,432**	**0**	**0**
Capitalized												
Capitalized Operating												
Till Removal	0	0	0	0	0	0	0	0	0	0	0	0
Mine Development	703	688	908	700	993	728	637	732	1,103	765	943	568
Capitalized	**703**	**688**	**908**	**700**	**993**	**728**	**637**	**732**	**1,103**	**765**	**943**	**568**
Working Capital												
Receivables	2,266	2,236	2,407	2,306	1,999	2,024	2,114	2,462	2,558	2,489	2,156	2,196
Payables	(4,501)	(4,406)	(4,331)	(4,002)	(4,234)	(4,216)	(4,358)	(4,242)	(4,328)	(4,353)	(3,998)	(3,865)
Working Capital	**2,235**	**2,170**	**1,924**	**1,696**	**2,235**	**2,192**	**2,244**	**1,780**	**1,770**	**1,864**	**1,842**	**1,669**
Change in Working Capital	(545)	65	246	229	(539)	43	(52)	464	10	(93)	22	172
TOTAL CAPITAL	**159**	**752**	**1,154**	**929**	**454**	**771**	**585**	**1,195**	**1,114**	**2,104**	**965**	**740**
Gross Income	**27,520**	**27,155**	**29,238**	**28,012**	**24,283**	**24,584**	**25,677**	**29,900**	**31,072**	**30,234**	**26,186**	**26,676**
Costs												
Operating	18,207	17,822	17,517	16,183	17,129	17,054	17,629	17,153	17,502	17,602	16,167	15,631
Capital	703	688	908	700	993	728	637	732	1,103	2,197	943	568
Working Capital	(545)	65	246	229	(539)	43	(52)	464	10	(93)	22	172
Total Costs	**18,366**	**18,575**	**18,671**	**17,112**	**17,583**	**17,826**	**18,214**	**18,348**	**18,615**	**19,705**	**17,132**	**16,371**
Cumulative	322,009	340,583	359,254	376,366	393,949	411,775	429,989	448,337	466,952	486,658	503,789	520,161
CASH FLOW	**9,155**	**8,581**	**10,567**	**10,901**	**6,700**	**6,758**	**7,462**	**11,552**	**12,457**	**10,528**	**9,054**	**10,305**
Cumulative	114,402	122,983	133,550	144,451	151,151	157,909	165,371	176,923	189,380	199,908	208,962	219,268
Present Value	7,412	6,861	8,346	8,503	5,161	5,142	5,608	8,574	9,131	7,622	6,474	7,278
NPV	92,737	99,598	107,944	116,446	121,608	126,750	132,357	140,931	150,062	157,685	164,159	171,437
IRR												

Description	2015 Q1 27	Q2 28	Q3 29	Q4 30	2016 Q1 31	Q2 32	Q3 33	Q4 34	2017 Q1 35	Q2 36	Q3 37	Q4 38	2018 Q1 39	Q2 40	Q3 41	Q4 42
CASH FLOW (Continued)																
Capital Costs																
Open Pit Mine	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
Underground Mine	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
Stock Mill & Tailings	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
Black Fox Mill	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
Infrastructure	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
Bridge to Waste Dump	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
Black FoxTailings Dam	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
Closure Costs	0	0	0	0	0	0	0	0	0	0	13,718	791	791	791	0	0
Owner Costs	0	0	0	0	0	0	0	0	0	0	0	4	0	0	0	0
Capital Costs	**0**	**0**	**0**	**0**	**0**	**0**	**0**	**0**	**0**	**0**	**13,718**	**795**	**791**	**791**	**0**	**0**
Capitalized																
Capitalized Operating																
Till Removal	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
Mine Development	538	585	239	237	124	0	0	0	0	0	0	0	0	0	0	0
Capitalized	**538**	**585**	**239**	**237**	**124**	**0**	**0**	**0**	**0**	**0**	**0**	**0**	**0**	**0**	**0**	**0**
Working Capital																
Receivables	2,346	1,974	1,993	1,787	1,528	1,267	1,394	1,556	1,249	1,223	515	0	0	0	0	0
Payables	(3,544)	(3,201)	(2,960)	(2,729)	(2,381)	(1,946)	(1,944)	(1,902)	(1,779)	(1,766)	(1,125)	0	0	0	0	0
Working Capital	**1,198**	**1,227**	**966**	**942**	**853**	**678**	**551**	**346**	**530**	**544**	**610**	**0**	**0**	**0**	**0**	**0**
Change in Working Capital	471	(28)	260	24	89	175	128	204	(183)	(14)	(66)	610	0	0	0	0
TOTAL CAPITAL	**1,009**	**557**	**499**	**261**	**213**	**175**	**128**	**204**	**(183)**	**(14)**	**13,651**	**1,405**	**791**	**791**	**0**	**0**
Gross Income	**28,493**	**23,979**	**24,211**	**21,703**	**18,556**	**15,391**	**16,926**	**18,891**	**15,170**	**14,846**	**6,246**	**0**	**0**	**0**	**0**	**0**
Costs																
Operating	14,325	12,940	11,961	11,031	9,624	7,862	7,854	7,679	7,185	7,134	4,545	0	0	0	0	0
Capital	538	585	239	237	124	0	0	0	0	0	13,718	795	791	791	0	0
Working Capital	471	(28)	260	24	89	175	128	204	(183)	(14)	(66)	610	0	0	0	0
Total Costs	**15,334**	**13,497**	**12,460**	**11,291**	**9,836**	**8,037**	**7,981**	**7,883**	**7,002**	**7,120**	**18,196**	**1,405**	**791**	**791**	**0**	**0**
Cumulative	535,495	548,992	561,452	572,743	582,579	590,616	598,597	606,481	613,482	620,602	638,798	640,203	640,994	641,785	641,785	641,785
CASH FLOW	**13,159**	**10,482**	**11,751**	**10,412**	**8,720**	**7,354**	**8,944**	**11,008**	**8,168**	**7,726**	**(11,950)**	**(1,405)**	**(791)**	**(791)**	**0**	**0**
Cumulative	232,426	242,909	254,660	265,072	273,792	281,146	290,090	301,098	309,266	316,992	305,042	303,637	302,846	302,055	302,055	302,055
Present Value	9,178	7,221	7,995	6,997	5,788	4,821	5,790	7,039	5,158	4,819	(7,362)	(855)	(475)	(469)	0	0
NPV	180,615	187,836	195,831	202,828	208,615	213,436	219,226	226,265	231,423	236,242	228,881	228,026	227,551	227,081	227,081	227,081
IRR																

17 Additional Requirements for Development Properties and Production (Item 25)

There is no known current information or data that could provide additional useful insight to Black Fox. All known relevant data and information is presented within this Technical Report and in the appropriate sections.

18 Interpretation and Conclusions (Item 21)

18.1 Interpretation

The Black Fox property is located within Precambrian age metavolcanics and metasedimentary rocks of the Abitibi Greenstone Belt. This is one of the worlds largest Archean greenstone belts believed to have formed by a complex history of paired arc volcanism and back arc sediments subsequently deformed during continental collision. The Black Fox mineralization is controlled by the Destor-Porcupine Fault Zone (DPFZ), a major, east-west trending, deep-seated, crustal fault zone. The DPFZ and its numerous splays are associated with many past and current producing gold mines and gold deposits in the Porcupine Camp. The deposits are located closest to Black Fox each host approximately the 800k to 1Moz of gold. The Dome-Hoyle Pond deposits located 65km west, have shown that gold bearing structures can be traced to 1,600m below surface where they remain open at depth. The Holt-Holloway Mine, located approximately 45km to the east has been developed down to 1,200m below surface.

The Black Fox deposit has currently been drill tested to 700m below surface where portions remain open at depth. The drilling density is sufficient to estimate mineral resources classified as indicated and inferred categories. The deposit contains 1,889 drillholes drilled from surface and underground totaling to 335,983m. QA/QC procedures were reviewed by an outside consultant and determined to be appropriate. Mineralization occurs as two main ore types. The “Flow Zones” show good geologic and grade continuity forming distinct lens shaped bodies, in contrast to the “Main Zone” where mineralization occurs as discontinuous stockwork zones defining smaller pods of mineralization. Each of these mineralization types have been modeled with unique estimation procedures.

18.1.1 Opportunities

The Black Fox deposit contains a significant gold Resource, parts of which are potentially exploitable by open pit and underground mining methods. The deposit is currently drill defined to approximately 700m below surface where portions remain unconfined. Black Fox is located midway between the Dome-Hoyle Pond and Holt-Holloway Mines, each of these have been developed and exploited to approximately 1,000m below surface. At the Dome-Hoyle Pond deposit, gold mineralization has been traced to a depth of 1,600m where it remains unconfined. The Black Fox deposit provides opportunity to be developed into a profitable gold mine with potential at depth for additional Resources.

The ability to mine the Black Fox orebody using open pit mining methods will dramatically increase the knowledge base surrounding the “Nuggety” nature of the orebody. In effect, a very large bulk sample will be taken which will quantify perceived assay problems associated with interpreting grade from small drillhole diameters. Since historical underground mining has suggested a significant, “mine call factor” the true quantity of contained gold ounces may increase significantly. An increase in contained gold will increase ore tonnages to extend mine life and/or allow stockpiles, which can be drawn from during high strip ratio mining periods. The lessons learned from open pit mining will also give confidence to underground operations and associated stope layout.

The cut and fill stopes are designed based on a 3m lift height. During the detailed design phase, there is an opportunity to optimize this stope height and to possibly mine to different design parameters in different areas of the orebody. In areas where the orebody is wide, the stope height could be increased to between 4m and 5m to improve productivity and reduce the unit cost per tonne. An increase in the stope height would also increase the ore to waste ratio thus reducing the amount of waste to be hauled to surface.

There is a significant opportunity to mine a greater proportion of the orebody than has been presented in this design. A long-term gold price of $650/oz has been used to calculate cut-off grade and hence the stoping limits. However, the current gold price is significantly higher and if used to plan the short-term production, would result in a cut-off grade of approximately 2gpt. This would lead to more material being included within the economic stope limits.

In the preparation of the ore reserve estimate, all inferred material had its grade zeroed and was in effect treated as waste dilution. In the current underground reserve design, approximately 100,000t of inferred material at 12gpt was set to zero grade but is included in the design.

18.1.2 Risks

The gold mineralization at Black Fox is relatively high grade and sporadic in distribution. This style of mineralization is common to many similar deposits both within the district and through out the world. Many of these have been exploited successfully by techniques unique to each deposit. At Black Fox, the near term risk resides in refining the appropriate development and exploitation techniques required to successfully accommodate the nature of the gold mineralization. Conversion of Inferred Mineral Resources to the Indicated category has proven to be of relatively low risk so far. A recently completed drilling program conducted by Apollo during 2007 tested 14% of the previously classified Inferred Resource. The results of the updated Resource estimation showed that 84% of the tested material was converted to the Indicated category and the remaining 16% was invalidated.

The geological model is a conservative estimate of grade and is the basis for the pit optimization, pit design and production schedule. Given the “Nuggety” nature of the deposit, the ratio of ore to waste governs fleet utilization assuming a consistent production rate. If more ore is actually mined than predicted in the model, less waste stripping will be required and equipment utilization will decrease.

Much of the bedrock material is known to contain arsenic, which can be leached when exposed to oxygen and water. While treatment plans have been designed, additional pre-emptive dump sequencing may be employed to encapsulate the “Dirty” rock increasing complexity in the dump scheduling of the deposit.

The pit is very close to a major highway and private properties. This may create noise, vibration, dust and generally affect the local area and the mining operations if any of these aspects become excessive.

The permitting schedule is a principal driver of the production schedule and while there is about 18 months of CEAA process to go through of flexibility in commencing phase 2 and 3, any more delay will mean trucks will be parked until the relevant permits are in place.

The overburden material is known to have poor geotechnical characteristics. Careful analysis of the “Till” face needs to be made on the NW corner of the pit close to the highway. If this area presents geotechnical stability issues, a retaining wall may need to be considered.

There is some noted concern about the geotechnical stability of the waste dump if it encroaches on “Till” material. If it were found that 10:1 slopes are required for these areas, the most practical solution would be to remain within the stable dump footprint and make the waste dump higher. This would affect haul costs.

The open-pit comes within 15m of underground workings when mining phase 1. Care will be needed when blasting around this area as the underground workings in question will be the major haul route until an additional decline is constructed.

The costs for mining have been inflating at a rapid rate over a relatively short time span. Increased costs will adversely affect what material can be considered to be economic and hence also affect the quantity of contained reserves.

Historically, underground grade control has been challenging due to the complex nature of the mineralization. These challenges are expected to continue as mining progresses deeper. Geologists will have to visit the faces after each blast to mark up the ore contacts and to determine if faces should be advanced.

In the current market it is often difficult to attract skilled mining, technical, supervisory and management personnel to an operation. However, the Black fox Project is geographically located in a historical mining camp area in close proximity to a major town with a number of current operating mines. As such there should not be great difficulty in hiring suitably trained and qualified personnel.

As with all underground mining operations, ground conditions pose a significant risk. There is enough experience of mining in the upper areas to be confident that these risks are not excessive. The most effective defense against risk from poor ground conditions is sound engineering practice, good miner training, well-motivated supervision and a high degree of management focus on safety and standards.

18.2 Conclusions

The Black Fox deposit has been adequately drill tested to estimate grade and tonnes classified as Indicated and Inferred Resources. The estimation results have defined an Indicated Resource potentially exploitable by open pit mining at a 1gpt-Au cut-off, of 4.8Mt with an average grade of 5.3gpt-Au containing 0.8Moz of gold. Additionally, it defines an Indicated Resource potentially exploitable by underground mining at a 3gpt-Au cut-off, of 1.7Mt with an average grade of 11.4gpt-Au containing 0.6Moz of gold.

The open pit and underground mine design have defined a combined Indicated ore reserve of 6.5Mt at 6.4gpt-Au for 1.3Moz of gold.

The Feasibility Study demonstrates that the project is technically feasible and has a robust economic performance with the design and operating criteria used and the assumed gold price projections.

A key factor to the robust economic performance is the recent agreement to acquire the existing Stock Mill at a substantial discount to the cost of a new mill, notwithstanding the requisite permitting and equipment lead times for the construction of a new mill.

19 Recommendations (Item 22)

Black Fox should continue to be developed to the detailed engineering level. The following recommendations for the project should be considered by Apollo:

- Continue to core drill specific areas of the ore body to further upgrade and extend the geological modeling for the project;
- Complete Stock Mill tailings testwork;
- Establish optimal Stock Mill capacity;
- Complete detailed engineering design work in all areas; and
- Refine the project implementation schedule.

Estimated cost for these recommendations is US$3.0million.

20 References (Item 23)

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21 Glossary

21.1 Mineral Resources and Reserves

21.1.1 Mineral Resources

The mineral resources and mineral reserves have been classified according to the "CIM Standards on Mineral Resources and Reserves: Definitions and Guidelines" (November 2005). Accordingly, the Resources have been classified as Measured, Indicated or Inferred, the Reserves have been classified as Proven, and Probable based on the Measured and Indicated Resources as defined below.

A Mineral Resource is a concentration or occurrence of natural, solid, inorganic or fossilized organic material in or on the Earth's crust in such form and quantity and of such a grade or quality that it has reasonable prospects for economic extraction. The location, quantity, grade, geological characteristics and continuity of a Mineral Resource are known, estimated or interpreted from specific geological evidence and knowledge.

An 'Inferred Mineral Resource' is that part of a Mineral Resource for which quantity and grade or quality can be estimated on the basis of geological evidence and limited sampling and reasonably assumed, but not verified, geological and grade continuity. The estimate is based on limited information and sampling gathered through appropriate techniques from locations such as outcrops, trenches, pits, workings and drillholes.

An 'Indicated Mineral Resource' is that part of a Mineral Resource for which quantity, grade or quality, densities, shape and physical characteristics can be estimated with a level of confidence sufficient to allow the appropriate application of technical and economic parameters, to support mine planning and evaluation of the economic viability of the deposit. The estimate is based on detailed and reliable exploration and testing information gathered through appropriate techniques from locations such as outcrops, trenches, pits, workings and drillholes that are spaced closely enough for geological and grade continuity to be reasonably assumed.

A 'Measured Mineral Resource' is that part of a Mineral Resource for which quantity, grade or quality, densities, shape, physical characteristics are so well established that they can be estimated with confidence sufficient to allow the appropriate application of technical and economic parameters, to support production planning and evaluation of the economic viability of the deposit. The estimate is based on detailed and reliable exploration, sampling and testing information gathered through appropriate techniques from locations such as outcrops, trenches, pits, workings and drillholes that are spaced closely enough to confirm both geological and grade continuity.

21.1.2 Mineral Reserves

A Mineral Reserve is the economically mineable part of a Measured or Indicated Mineral Resource demonstrated by at least a Preliminary Feasibility Study. This Study must include adequate information on mining, processing, metallurgical, economic and other relevant factors that demonstrate, at the time of reporting, that economic extraction can be justified. A Mineral Reserve includes diluting materials and allowances for losses that may occur when the material is mined.

A 'Probable Mineral Reserve' is the economically mineable part of an Indicated, and in some circumstances a Measured Mineral Resource demonstrated by at least a Preliminary Feasibility Study. This Study must include adequate information on mining, processing, metallurgical, economic, and other relevant factors that demonstrate, at the time of reporting, that economic extraction can be justified.

A 'Proven Mineral Reserve' is the economically mineable part of a Measured Mineral Resource demonstrated by at least a Preliminary Feasibility Study. This Study must include adequate information on mining, processing, metallurgical, economic, and other relevant factors that demonstrate, at the time of reporting, that economic extraction is justified.

21.2 Glossary

Table 21.2.1: Definitions of Terms

Term	Definition
Assay:	The chemical analysis of mineral samples to determine the metal content.
Composite:	Combining more than one sample result to give an average result over a larger distance.
Concentrate:	A metal-rich product resulting from a mineral enrichment process such as gravity concentration or flotation, in which most of the desired mineral has been separated from the waste material in the ore.
Crushing:	Initial process of reducing ore particle size to render it more amenable for further processing.
Cut-off Grade (CoG):	The grade of mineralized rock, which determines as to whether or not it is economic to recover its gold content by further concentration.
Dilution:	Waste, which is unavoidably mined with ore.
Dip:	Angle of inclination of a geological feature/rock from the horizontal.
Fault:	The surface of a fracture along which movement has occurred.
Flow Ore:	A medium to fined grained, basal mafic volcanic rock which is generally located along the footwall of the deposit.
Footwall:	The underlying side of an ore body or stope.
Grade:	The measure of concentration of gold within mineralized rock.
Hangingwall:	The overlying side of an ore body or slope.
Level:	Horizontal tunnel the primary purpose is the transportation of personnel and materials.
Lithological:	Geological description pertaining to different rock types.
NQ Size:	A letter name specifying the dimensions of bits, core barrles, and drill rods in the N-size and Q-group wireline diamond drilling system having a core diameter of 47.6mm and a hole diameter of 75.7mm.
Ongoing Capital:	Capital estimates of a routine nature which is necessary for sustaining operations.
Operating Costs:	Sum of cost of mining, beneficiation, and administration gives the operating cost of the mine.
Ore Reserve:	See Mineral Reserve.
Sedimentary:	Pertaining to rocks formed by the accumulation of sediments, formed by the erosion of other rocks.
Sill:	A thin, tabular, horizontal to sub-horizontal body of igneous rock formed by the injection of magma into planar zones of weakness.
Specific Gravity:	The weight of a substance compared with the weight of an equal volume of pure water at 4ºC.
Stope:	Underground void created by mining.
Strike:	Direction of line formed by the intersection of strata surfaces with the horizontal plane, always perpendicular to the dip direction.
Sulfide	A sulfur bearing mineral.
Variogram:	A statistical representation of the characteristics (usually grade)

Abbreviations

The metric system has been used throughout this report unless otherwise stated. All currency is in U.S. dollars. Market prices are reported in US$ per troy oz of gold and silver. Tonnes are metric of 1,000kg, or 2,204.6lbs. The following abbreviations are used in this report.

Table 21.2.2: Abbreviations of Units and Terms

Abbreviation	Unit or Term
AA	Atomic Absorption
ABA	Acid Base Analysis
ADR	Adsorption-Desorption-Recovery
amsl	Above mean sea level
AMEC	Association of Mining & Exploration Companies
ANFO	Ammonium Nitrate Fuel Oil (explosive)
Au	Gold
AUV	Ankerite Ultramafic
BMV	Bleached Mafic Volcanic
ºC	Degrees Centigrade
C$	Canadian dollar
CEAA	Canadian Environmental Assessment Act
CGR	Green Carbon Schists
CGY	Grey Carbonate
CIP	Carbon-in-pulp
cm	Centimeter
CoG	Cut-off-Grade
CUV	Chlorite-talc Ultramafic
°	Degree (degrees)
dia.	Diameter
DPFZ	Destor-Porcupine Fault Zone
FI	Felsic Intrusive
ft	Foot (feet)
ft^2	Square Foot (feet)
g	Gram
gal	Gallon
g/hr	Grams per hour
g/L	Grams per Liter
g/yr	Grams per year
gpt	Grams Per Tonne
G&A	General & Administration
ha	Hectares
hp	Horse Power
hr	Hour
IP	Induced Polarization
IRR	Internal Rate of Return
k	Thousand
km	Thousand Meters
koz	Thousand Troy Ounces
kt	Thousand Tonnes
kt/yr	Thousand Tonnes per Year
lb	Pound
LHD	Load Haul Dump
LoM	Life-of-Mine
m	Meter
m^2	Square meters
m^3	Cubed meters
MCC	Motor Control Center

Table 21.2.2: Abbreviations of Units and Terms (Continued)

Abbreviation	Unit or Term
MDA	Mine Development Associates
min	Minute
μm	Micron
mm	Millimeter
MNDM	Ministry of Northern Development and Mines
MOE	Ministry of the Environment
MOU	Memorandum of Understanding
Moz	Million troy ounces
Mt	Million tonnes
MTO	Ministry of Transportation
MV	Mafic Volcanic
NaCN	Sodium Cyanide
NGO	Non-government Organizations
NPV	Net Present Value
O&M	Operating & Maintenance
oz	Ounce
PMV	Pillowed Mafic Volcanic
ppm	Parts per Million
%	Percent
QA/QC	Quality Assurance/Quality Control
RMSB	Ross Mine Syenitic Belt
RoM	Run-of-Mine
RPA	Scott Wilson Roscoe Postle Associates Inc.
SED	Lens of Greywacke
SUV	Silicified Grey Carbonate
t	Tonne (metric ton) (2,204.6 pounds)
tpd	Tonnes per day
tph	Tonnes per hour
tpy	Tonnes per year
TUV	Talc Ultramafic
UTEM	University of Toronto ElectroMagnetometer
yr	Year

Appendix A

Certificates of Author

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USA 80235
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CERTIFICATE of AUTHOR

I, Bart A. Stryhas Ph.D. CPG # 11034 do hereby certify that:

1. I am a Principal Resource Geologist of:

   SRK Consulting (US), Inc.
   7175 W. Jefferson Ave, Suite 3000
   Denver, CO, USA, 80235

2. I graduated with a Doctorate degree in structural geology from Washington State University in 1988. In addition, I have obtained a Master of Science degree in structural geology from the University of Idaho in 1985 and a Bachelor of Arts degree in geology from the University of Vermont in 1983.

3. I am a current member of the American Institute of Professional Geologists.

4. I have worked as a Geologist for a total of 20 years since my graduation in minerals exploration, mine geology, project development and resource estimation. I have conducted resource estimations since 1988 and have been involved in technical reports since 2004.

5. I have read the definition of "qualified person" set out in National Instrument 43-101 ("NI 43-101") and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101.

6. I am responsible for the entire report excluding the Social and Environmental, Geotechnical, Mineral Processing and Metallurgical Testing and Mining and Reserves considerations part of the technical report, titled, NI 43-101 Technical Report, Apollo Gold Corporation, Black Fox Project, and dated April 14, 2008 (the "Technical Report") relating to the Black Fox property. I visited the Black Fox property for two days on October 2 and 3, 2007.

7. I have had prior involvement with the property that is the subject of the Technical Report.

8. As of the date of the certificate, to the best of the qualified person's knowledge, information and belief, the technical report contains all scientific and technical information that is required to be disclosed to make the technical report not misleading.

9. I am independent of the issuer applying all of the tests in Section 1.4 of National Instrument 43-101.

10. I have read National Instrument 43-101 and Form 43-101F1, and the Technical Report has been prepared in compliance with that instrument and form.

11. I consent to the filing of the Technical Report with any stock exchange and other regulatory authority and any publication by them for regulatory purposes, including electronic publication in the public company files on their websites accessible by the public, of the Technical Report.

Dated April 14, 2008.

____

Dr. Bart A. Stryhas, CPG, PhD (Signed)

SRK Consulting
Engineers and Scientists

SRK Consulting (U.S.), Inc.
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CERTIFICATE of AUTHOR

I, Martin Raffield, P.Eng., do hereby certify that:

1. I am a Senior Mining Engineer of:

   SRK Consulting (US), Inc.
   7175 W. Jefferson Ave, Suite 3000
   Denver, CO, USA, 80235

2. I graduated with a Doctorate degree in mining engineering from Cardiff University in 1993. In addition I have obtained a Bachelor of Science degree in mining geology from Cardiff University in 1989.

3. I am a licensed professional engineer registered with Professional Engineers Ontario (PEO).

4. I have worked as a mining engineer for a total of 14 years since my graduation from university.

5. I have read the definition of "qualified person" set out in National Instrument 43-101 ("NI 43-101") and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101.

6. I am responsible for the preparation of sections 16.1 and 16.2 of the technical report titled NI 43-101 Technical Report, Apollo Gold Corporation, Black Fox Project, Timmins, Ontario, Canada and dated April 14, 2008 (the "Technical Report"), relating to the Black Fox property. I visited the Black Fox property on Nov 1, 2007.

7. I have had prior involvement with the property that is the subject of the Technical Report.

8. I am independent of the issuer applying all of the tests in section 1.4 of National Instrument 43-101.

9. I have read National Instrument 43-101 and Form 43-101F1, and the Technical Report has been prepared in compliance with that instrument and form.

10. As of the date of this certificate, to the best of my knowledge, information and belief, the Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated this 14th Day of April, 2008.

Dr Martin P. Raffield, P.Eng., PhD. (Signed) PEO License No.: 100061761 (Sealed)

CERTIFICATE of AUTHOR

I, Debbie Dyck, P. Eng., do hereby certify that:

1. I am an Associate Environmental Engineer of:

   AMEC Earth & Environmental
   a division of AMEC Americas Limited
   160 Traders Blvd. E., Suite 110
   Mississauga, Ontario, Canada, L4Z 3K7

2. I graduated with a degree in chemical engineering (B.A.Sc.) from the University of Waterloo, Ontario in 1990.

3. I am a member of the Association of Professional Engineers of Ontario.

4. I have worked as an engineer for over 17 years since my graduation from university.

5. I have read the definition of "qualified person" set out in National Instrument 43-101 ("NI 43-101") and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101.

6. I am responsible for the preparation of Sections 2.5 and 16.8 of the technical report titled "*NI 43-101 Technical Report, Apollo Gold Corporation, Black Fox Project, Timmins, Ontario, Canada*" and dated April 14, 2008 (the "Technical Report") relating to the Black Fox property. I have visited the Black Fox property on several occassional over the past five years.

7. I have had prior involvement with the Black Fox property that is the subject of the Technical Report. I was involved with the preparation and submission of the Closure Plan for the existing site conditions at the Black Fox property on behalf of Apollo Gold Mines Ltd. In addition, I have overseen the geotechnical, geochemical, hydrological, and environmental studies completed by AMEC to date.

8. I am independent of the issuer applying all of the tests in section 1.4 of National Instrument 43-101.

9. I have read National Instrument 43-101 and Form 43-101F1, and the Technical Report sections that I prepared have been prepared in compliance with that instrument and form.

10. I consent to the filing of the Technical Report with any stock exchange and other regulatory authority and any publication by them for regulatory purposes, including electronic publication in the public company files on their websites accessible by the public, of the Technical Report.[1]

11. As of the date of this certificate, to the best of my knowledge, information and belief, the Technical Report sections that I prepared contain all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

[1]If an issuer is using this certificate to accompany a technical report that it will file only with the exchange, then the exchange recommends that this paragraph is included in the certificate.

Dated this 11th Day of April, 2008.

LICENSED PROFESSIONAL ENGINEER
C. D. DYCK
11/04/08
PROVINCE OF ONTARIO

Debbie Dyck, P.Eng.

(Sealed)

CERTIFICATE of AUTHOR

I, Xiaogang Hu, P. Eng., do hereby certify that:

1. I am a Senior Associate Engineer of:

   AMEC Earth & Environmental
   a division of AMEC Americas Limited
   160 Traders Blvd. E., Suite 110
   Mississauga, Ontario, Canada, L4Z 3K7

2. I graduated with a degree in Hydrology (B.Eng.) from the Chengdu University of Science and Technology, Chengdu, China, in 1982. In addition, I have obtained a Ph.D. in Northern Hydrology from McGill University, Montreal, Quebec, in 1997.

3. I am a member of the Association of Professional Engineers of Ontario.

4. I have worked as an engineer for over 25 years since my graduation from university.

5. I have read the definition of "qualified person" set out in National Instrument 43-101 ("NI 43-101") and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101.

6. I am responsible for the preparation of Sections 16.1.20 and 16.3 of the technical report titled "*NI 43-101 Technical Report, Apollo Gold Corporation, Black Fox Project, Timmins, Ontario, Canada*" and dated April 14, 2008 (the "Technical Report") relating to the Black Fox and Stock Mine properties.

7. I have not had prior involvement with the properties that are the subject of the Technical Report.

8. I am independent of the issuer applying all of the tests in section 1.4 of National Instrument 43-101.

9. I have read National Instrument 43-101 and Form 43-101F1, and the Technical Report sections that I prepared have been prepared in compliance with that instrument and form.

10. I consent to the filing of the Technical Report with any stock exchange and other regulatory authority and any publication by them for regulatory purposes, including electronic publication in the public company files on their websites accessible by the public, of the Technical Report.[1]

11. As of the date of this certificate, to the best of my knowledge, information and belief, the Technical Report sections that I prepared contain all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

[1]If an issuer is using this certificate to accompany a technical report that it will file only with the exchange, then the exchange recommends that this paragraph is included in the certificate.

Dated this 11th Day of April, 2008.

LICENSED PROFESSIONAL ENGINEER
X. G. HU
100057467
11/04/08
PROVINCE OF ONTARIO

Dr. Xiaogang Hu, P.Eng.

(Sealed)

CERTIFICATE OF AUTHOR

I Randolph P Schneider, MAusIMM (CP) do hereby certify that:

1. I am a Senior Process Engineer of:

   Samuel Engineering, Inc
   8450 E Crescent Pkwy, Suite 200
   Greenwood Village, CO 80111

2. I graduated with a degree in Metallurgical Engineering from the Colorado School of Mines in 1971.

3. I am a member and chartered professional of the AusIMM.

4. I have worked as a metallurgical engineer for a total of 34 years since my graduation from the Colorado School of Mines.

5. I have read the definition of "qualified person" set out in National Instrument 43-101 ("NI 43-101") and certify that by reason of education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101.

6. I am responsible for the Metallurgy, Processing and Infrastructure sections of the technical report titled NI 43-101 Feasibility Study, Apollo Gold Corporation, Black Fox, Timmins, Ontario, Canada and dated April 14, 2008 (the "Technical Report") relating to the Black Fox property.

7. I have had prior involvement with the property that is the subject of the Technical Report. The nature of my prior involvement was preparation of the metallurgical section for the August 2007 NI 43-101 Black Fox Prefeasibility Study.

8. I am not aware of any material fact or material change with respect to the subject matter of the Technical Report that is not reflected in the Technical Report, the omission to disclose which makes the Technical Report misleading.

9. I am independent of the issuer applying all of the tests in Section 1.5 of National Instrument 43-101.

10. I have read National Instrument 43-101 and Form 43-101F1, and the Technical Report has been prepared in compliance with that instrument and form.

11. I consent to the filing of the Technical Report with any stock exchange and other regulatory authority and any publication by them for regulatory purposes, including electronic publication in the public company files on their websites accessible by the public, of the Technical Report.

Dated this 14 April, 2008

Randolph P Schneider
(Signed)

NI 43-101 Technical Report, Apollo Gold Corporation, Black Fox Project, Timmins, Ontario, Canada dated this 29th Day of February 2008.

Dated this 14th Day of April 2008.

Bart Stryhas, CPG, PhD (signed)

Martin Raffield, P.Eng., PhD (signed)

Debbie Dyck, BASc, PEng (signed)

Randolph P. Schneider, MAusIMM (signed)

Xiaogang Hu, PhD, PEng (signed)